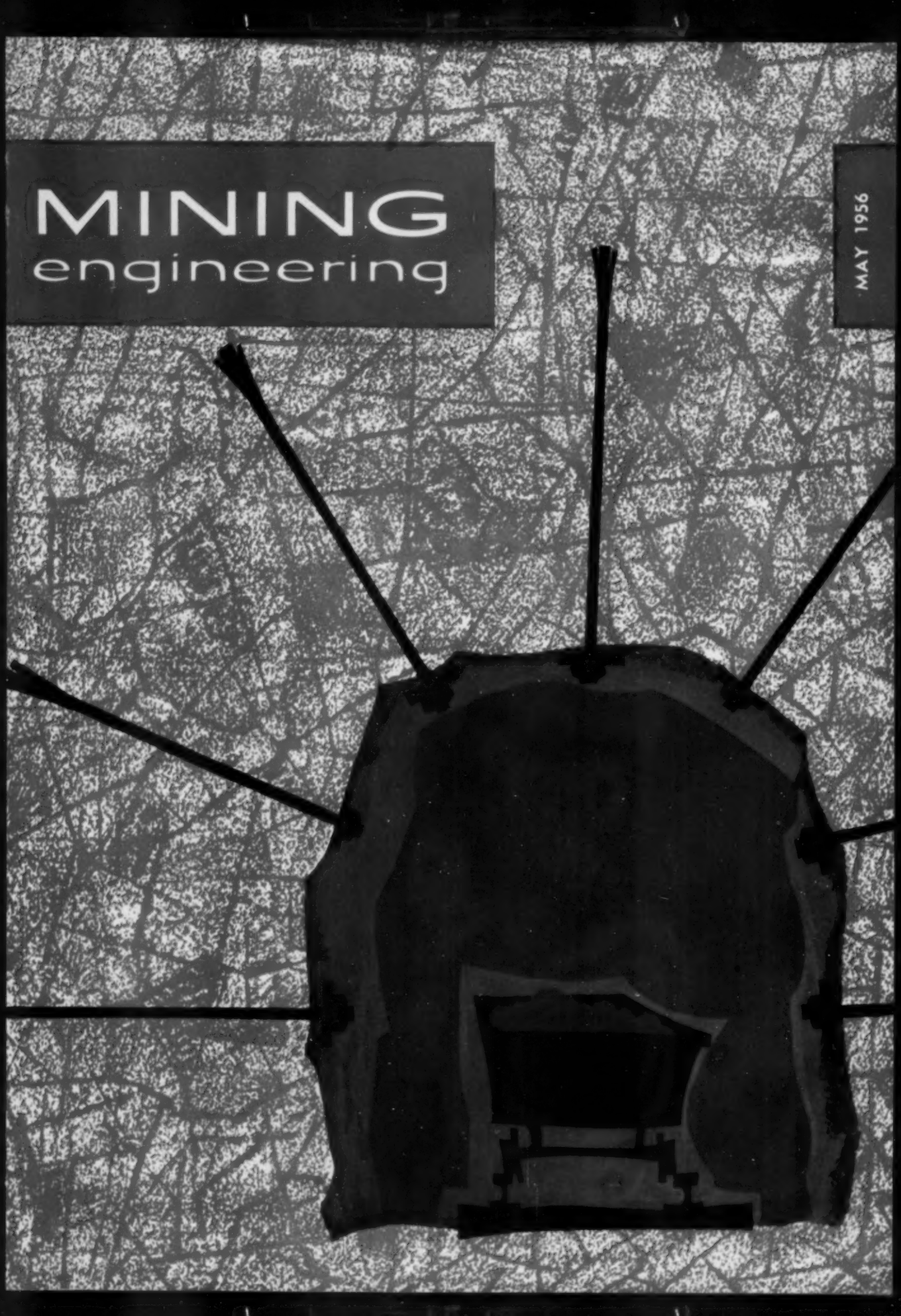
MINING engineering

MAY 1956



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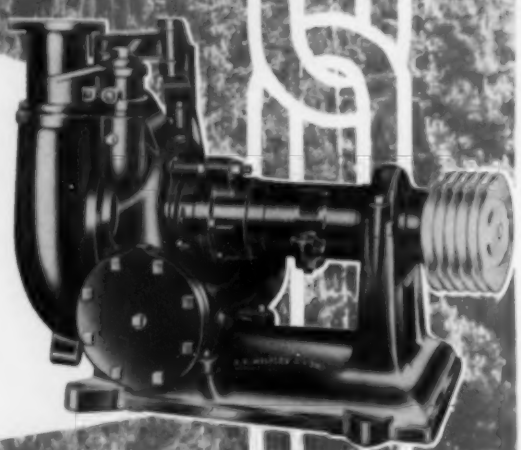

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MINING engineering

VOL. 8 NO. 5 MAY 1956

COVER

Radiating rock bolts mean more to the mining industry than an interesting pattern for Cover Artist Herb McClure. J. L. Humphrey in the article beginning on page 491 points out that proper application means lower costs with much greater safety than timber sets.

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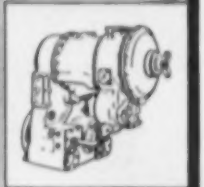

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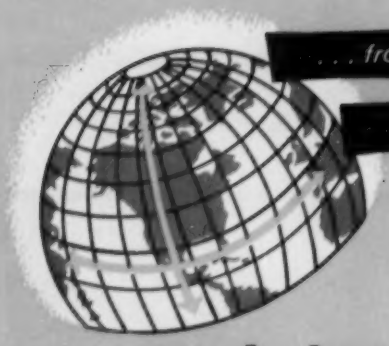

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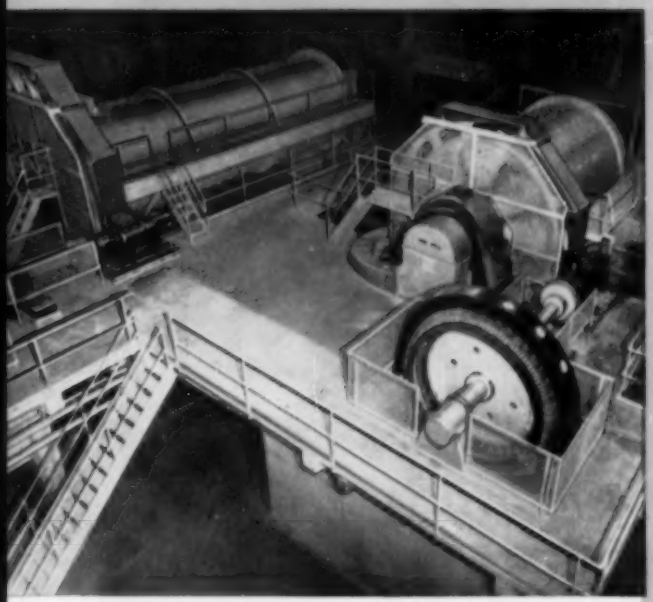
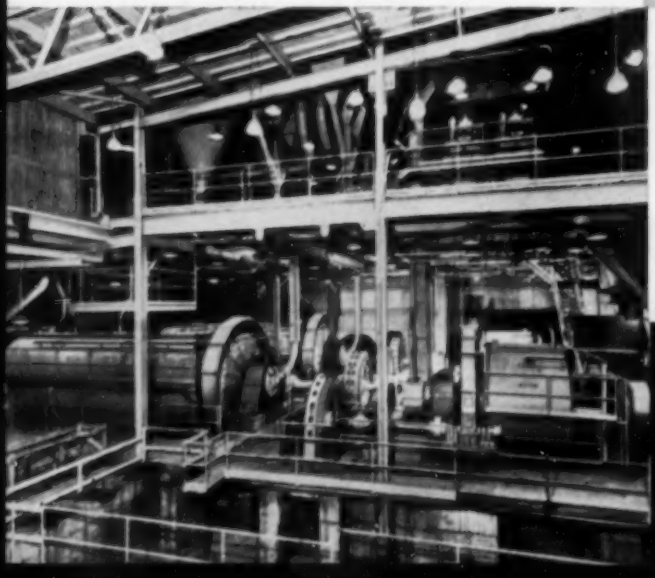


These two views show eight Nordberg Grinding Mills used for raw and finish grinding in the plants of a leading cement producer. Above is a 10'8" by 16' Ball Mill and a 9'6" by 32' Compartment Mill. Below are two 10'8" by 17' and two 9'6" by 11' Ball Mills, together with two 8'6" by 36' Compartment Mills.



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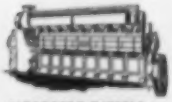

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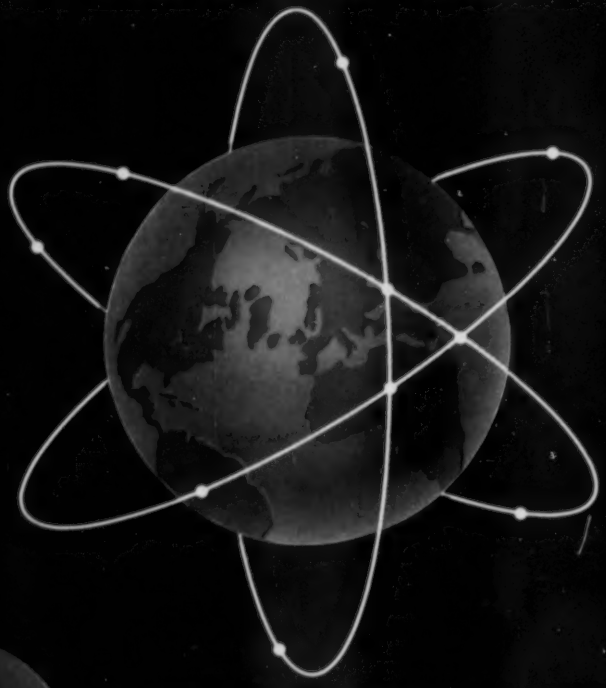

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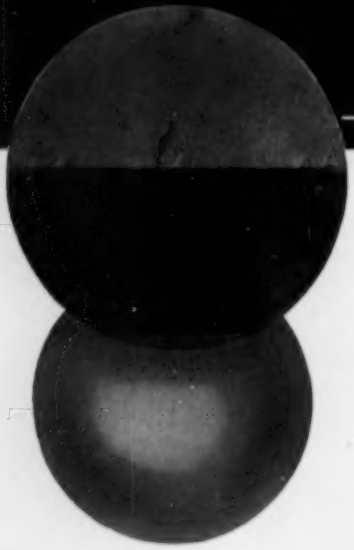

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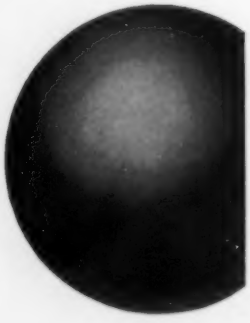

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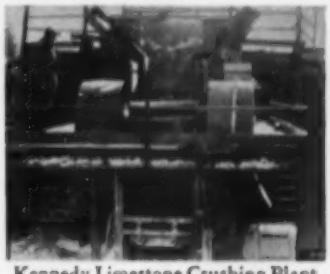

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Clays and Clay Minerals, Proceedings of the Second National Conference on Clays and Clay Minerals, edited by Ada Swineford and Norman Plummer, *National Academy of Sciences—National Research Council,* 2101 Constitution Ave., Washington 25, D. C., Publication 327, $4.00, 512 pp., 1954.—The conference was held at the University of Missouri, Oct. 15 to 17, 1953. The 36 papers are by 64 contributors whose fields included geology, mineralogy, soil chemistry, and soil mechanics, physics, colloid chemistry, and ceramics. Also given is a detailed and annotated mileage guide to the diaspore and flint clays and to the Putnam soil of Missouri, which were visited on a field trip. The atomic structure of the clay minerals was specially featured at the conference with particular attention to the chlorite and mixed-layer clay minerals.

Ohio Fossils, by Aurèle La Rocque and Mildred Fisher Marple, *Ohio Div. of Geological Survey,* Room 106, Orton Hall, Ohio State University, Columbus 10, Ohio, Bulletin 54, 97¢, 152 pp., 1955.—An introduction to the paleontology of Ohio written for the nonprofessional. Included are methods of collection, preparation, basic concepts of the Linnaean system, and a nontechnical geologic history of the state. Main features of the most abundant fossils are illustrated in 413 line drawings.

Chemical Engineering Processes and Equipment, edited by E. Molloy and E. Carr, *George Newnes Ltd.,* London. Available in U. S. from Transatlantic Arts, Inc., Hollywood-by-the-Sea, Fla., $5.00, 236 pp., 1956.—Concise descriptions of a wide range of processes and equipment are presented, supplemented in some cases with lists of references to more detailed information. The equipment dealt with includes ball and pebble mills, mixers, porous stainless steel filters, heat exchangers, film dryers, electrical precipitators, and automatic dust collectors. Among the processes covered are: vibro-agitation, tabling, magnetic separation, elutriation, evaporating, crystallization, and heavy media separation.

Rocks and Minerals, by Richard M. Pearl, Everyday Handbook Series, *Barnes & Noble Inc.,* 105 Fifth Ave., N. Y. 3, N. Y., $1.95, 275 pp., 1956.—Written in popular language for the general reader, this book offers the most recent knowledge about radioactive and fluorescent minerals, ores and metals, gems, meteorites, etc. Also included are the artificial minerals, from their origin to their current industrial uses. Classification of rocks and minerals is explained as well as ways to recognize, collect, and display them.

Minerals Yearbook, Fuels, 1953, Volume II, by the staff of the U. S. Bureau of Mines, Superintendent of Documents, *U. S. Government Printing Office,* Washington 25, D. C., $2.25, 478 pp., 1956.—This volume is devoted to the mineral fuels and consists of chapters on each mineral-fuel commodity, as well as chapters reviewing the industry as a whole. There is also a statistical summary and a presentation on employment and injury. The total new energy supply in the U. S. in 1953 increased more than 1 pct over 1952. Income from the crude petroleum and natural-gas industries continued to rise, but declined in the coal industries.

Modern Techniques of Excavation, by Herbert L. Nichols, Jr., *North Castle Books,* 212 Bedford Rd., Greenwich, Conn., $9.00, 608 pp., 1956.—This is the third in a practical series on earth-moving. The book provides information about the planning and execution of dirt and rock excavation and grading projects of all kinds, but does not include details of the construction and basic operation of equipment. Two chapters deal with financial management, estimating, insurance, and certain aspects of equipment maintenance. The glossary contains definitions of more than 1200 terms used in the industry.

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Analyse der Metalle, Volume III Probenahme, *Springer-Verlag,* Berlin, DM45, 452 pp. 1956.—The first section of this manual on the sampling of ores, metals, fuels, and other metallurgically important materials, deals with the fundamentals of sampling, the necessary apparatus, and standard procedures. The second section gives detailed information on the sampling of some 50 metals and metalloids from their ores to the finished product. It also contains facts about a number of fuels, refractories, fluxes, etc. The bibliography covers the pertinent literature from 1860 to 1954.

Basic Procedures of Diamond and Shot Core Drilling, *Acker Drill Co. Inc.,* Scranton, Pa., $1.00, 35 pp., 1956.—Though written in nontechnical language, this manual is based on much research and operational data on both diamond and shot core drilling. Chapters detail setting up the core drill, driving the casing, hints for successful drilling, etc.

The Analysis of Minerals and Ores of the Rarer Elements, by W. R. Schoeller and A. R. Powell, *Hafner Publishing Company,* 31 East 10th Street, N. Y. 3, N. Y., $9.75, 408 pp. 3rd edition, 1955.—General information on methods and apparatus,

(Continued on page 458)

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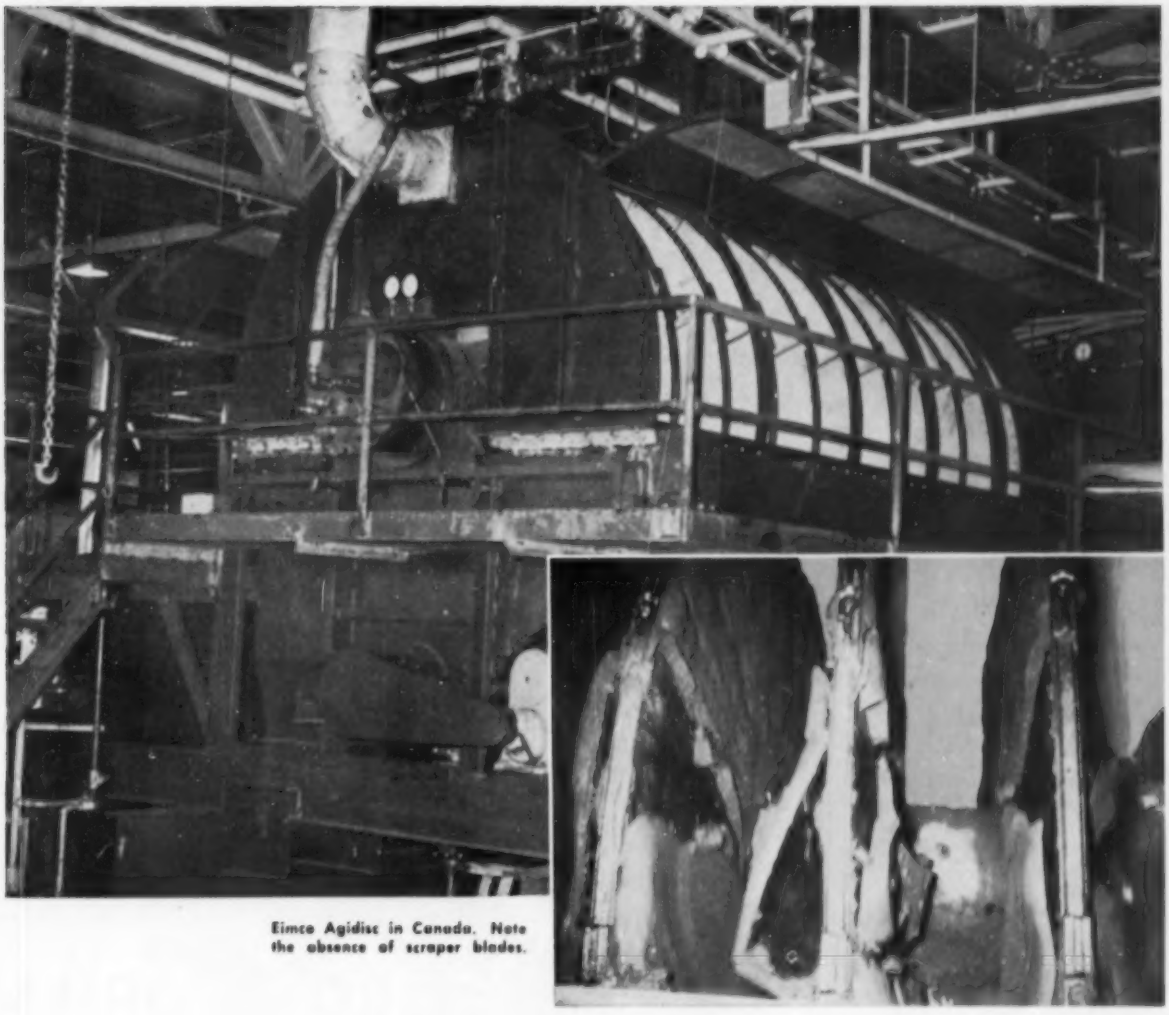

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B-104

BOOKS

(Continued from page 456)

and discussions of mineralogical, qualitative, and complete chemical analyses precede the main part of this book, which consists of separate chapters devoted to the analysis of each element. Usually given for each, are minerals, properties, compounds, methods of separation and determination, and methods of detection in minerals. Such new methods as chromatography for uranium, solvent extraction for scandium, and spectrophotometry for the rare earths have been included in this edition. A number of chapters have been revised or rewritten.

ORDER THE FOLLOWING BOOKS THROUGH AIME—Address Irene K. Sharp, Book Dept. Ten percent discount given whenever possible.

The Technology of Cement and Concrete: Volume I, Concrete Materials, by Robert F. Blanks and Henry L. Kennedy, *John Wiley & Sons Inc.*, $11.00, 422 pp., 1955.—Known facts about materials used in making concrete and an examination of their concrete-making qualities are given. With special attention to Portland cement, the authors have divided this volume into two sections. The first deals with Portland cements and their manufacture, pozzolanic cements, other hydraulic cements, and additions and admixtures. The section covering concrete aggregate materials includes: geology and petrology of concrete aggregates; effect of aggregate properties on concrete; prospecting, exploration, sampling, and testing for concrete aggregates. Volume II, now in preparation, will deal with the most effective combinations of concrete materials as well as the properties and uses of resulting concrete.

Microscopic Petrography by E. Wm. Heinrich, *McGraw-Hill Book Co. Inc.*, $6.50, 296 pp., 1956.—This book in the Geological Sciences Series, McGraw-Hill, is designed for practicing geologists, particularly in the mining industries, and students of geology and mineralogy. It offers a complete treatment of the microscopic investigation of igneous, sedimentary, and metamorphic rocks. Information about each rock described is given under various subheadings: definition, mineralogy, textures and microstructures, occurrence, and derivation. Other topics discussed are the use of insoluble residues from both carbonate and evaporite rocks, the use of heavy detrital minerals from clastic sedimentary rocks, and evaporite rocks.

The Metallurgy of Zirconium, edited by Benjamin Lustman and Frank Kerze, Jr., *McGraw-Hill Book Co. Inc.*, **Div. VII, vol. 4, National Nuclear Energy Series,** $10.00, 776 pp., 1956.—More than 50 specialists have prepared this book which is a complete report on the present state of zirconium, covering occurrence, reduction, refining, properties, fabrication, and uses. It will be of interest to research and operating engineers concerned with plants handling zirconium and its alloys. An outgrowth of the AEC and the U.S. Navy's effort to use atomic energy for marine propulsion, the book is particularly strong in its treatment of such topics as corrosion properties of zirconium, fabrication techniques, and the iodide method of refining.

Economic Geography of Industrial Materials, edited by Albert S. Carlson, *Reinhold Publishing Corp.*, $12.50, 494 pp., 1956.—Top experts in a wide range of key industries have contributed chapters to make this new handbook one that answers a great many perplexing problems facing industry today. Each chapter contains maps, tables, charts and statistics and describes the economic-geographic conditions necessary for profitable production. Among the factors discussed are location, climate, soil, topography, water resources, minerals, fuels, labor, capital, management, and transportation. These are also evaluated to show profitable trends and conditions at present and in the future.

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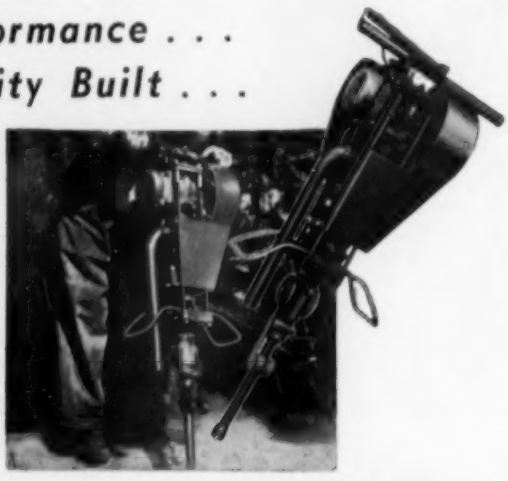

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The Syntron Rock Drill is a versatile, rugged, one-man operation. Easily portable over rough terrain—requires no auxiliary equipment. A complete drilling outfit for the very lowest cost.

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Traylor's Long Experience Building Gyratory Crushers is Your Short Cut to Profits

More than a half-century of experience is reflected in the outstanding design features of a Traylor TC Gyratory Crusher.

The proven production ability of a TC is due mainly to the unique design of its bell head and curved concaves. The design of Traylor original curved crushing surfaces utilizes the full force of the power applied to quickly crush the material as it enters the crushing chamber. And, because each zone in the crushing chamber has greater capacity than the preceding zone, choking seldom occurs.

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Tough Digging in Confined Areas

Eimco 105 Tractor Excavators can work efficiently in small areas because of their built in features.

These features permit the Eimco 105 to work in many places such as narrow ditch digging, tunnels, water channels, mines, railroad cuts, highway widening jobs and numerous others including basement excavation as shown above.

The Eimco 105 features include: (1) Independent track control so that one track can be run forward while the other runs reverse providing spin turns in the length of the machine, (2) Up front position of the operator for better visibility in confined areas, (3) Overhead bucket motion, to eliminate the necessity of turning to discharge the bucket, (4) Simple easy controls, two small handles are all that control the motion of the tractor. Push for forward motion and pull for reverse motion.

Eimco 105 Tractor-Excavators are being used on construction jobs all over the world as well as in mines and quarries. These machines are operating longer periods at lower cost per yard loaded than any other equipment.

Investigate the Eimco Tractor-Excavator — it is the world's finest crawler tractor equipment.

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B-105

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Gardner-Denver 125-foot portable at entrance to Utah uranium mine. Supplies air for drilling within the mine.

"Cash in" quicker on uranium with Gardner-Denver quality equipment

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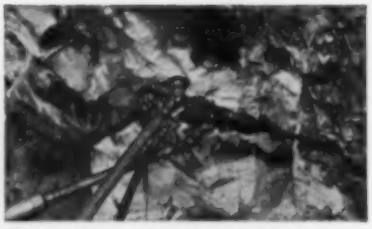
Send for bulletins on Gardner-Denver portable or skid-mounted compressors, drilling equipment for deep test holes, sinkers, drifters, stopers, air hoists, mine car loaders.



S-48 sinker and air feed leg — a lightweight, fast-drilling combination.

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CF&I Rock Bolts eliminate the need for cumbersome timber—give secure support to mine roofs and walls—reduce dangers of roof falls due to blast effect. Their use makes more working space available—permits better ventilation—increases efficiency of mine equipment and manpower.

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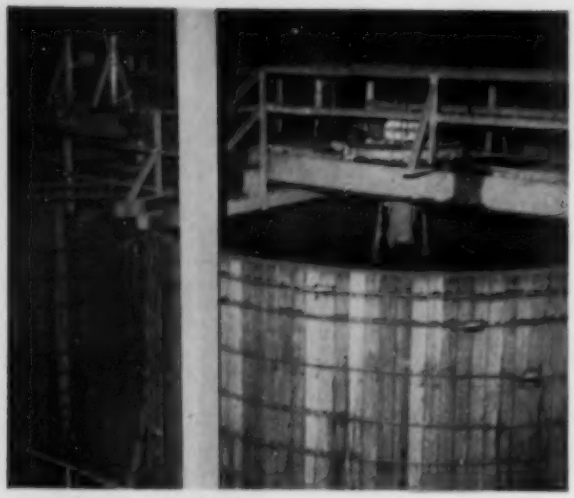
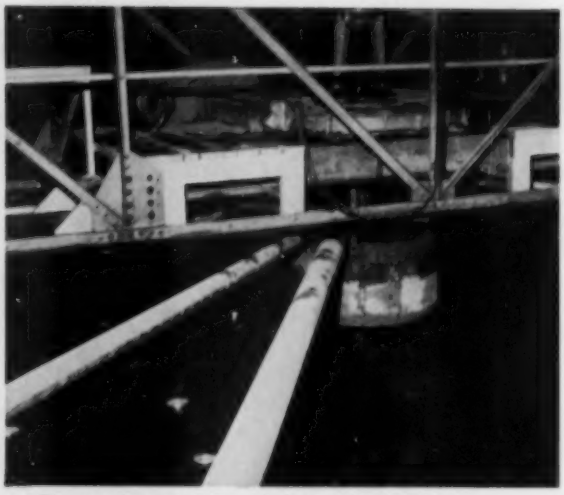
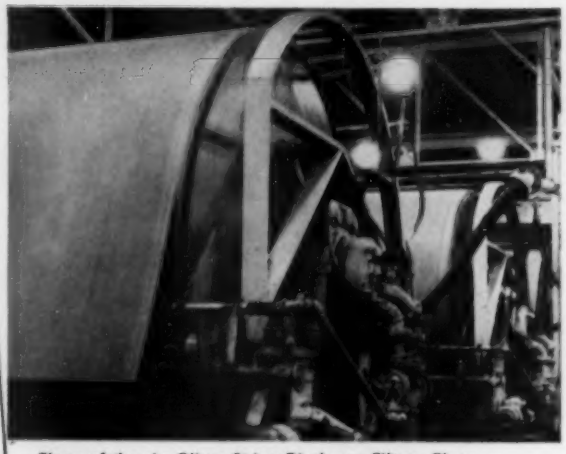


Two of the twelve Dorr Agitators installed at Gunnar.



Closeup of one of the Tray Thickeners showing drive mechanism and feeding arrangement.

URANIUM PRODUCTION AT GUNNAR MINES LTD.



Three of the six Oliver String Discharge Filters. Three additional Filters have since been ordered.

Photographs by J. Wilson, Gunnar Mines Ltd.

One of the early Uranium producers in North America, Gunnar Mines Limited at Uranium City, Saskatchewan selected Dorr-Oliver-Long equipment for above and below ground operations from our Associate Company in Canada. For the *leaching section* twelve acid resisting Dorr Agitators are installed, each 20′ dia. by 20′ deep. For the *neutral circuits* four 3-compartment Dorr Tray Thickeners are used, each 50′ dia. by 23′ deep and two 8′6″ dia. x 7 Disc American Filters. For *final residue washing*, six Oliver String Discharge Filters are employed, each 11′ 6″ x 16′ face with acid resisting construction. In the *precipitation section* is the first Rubber Lined Sweetland Filter ever built.

Now an open pit operation, Gunnar plans a new shaft in the near future and has ordered from Dorr-Oliver-Long the front dump skip, cage, head sheaves and Granby type mine cars.

For more information on the complete line of D-O equipment for the mining industry, write to Dorr-Oliver, Stamford, Connecticut.

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A NEW ERA IN UNDERGROUND MECHANIZATION

with the

EIMCO Crawler Model 630 Series

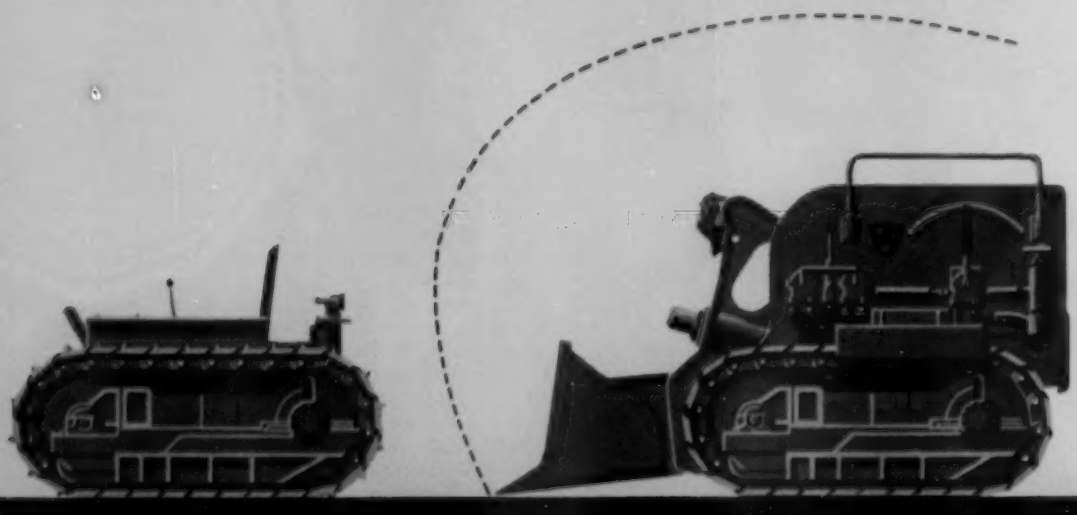
The Eimco 630 series has been developed, at the request of operators, to take numerous types of auxiliary equipment.

The basic tractor unit, with the excavator attachment, has been adequately field tested. No tractor unit is ever subjected to rigorous field testing until it has an overhead loader of the Eimco type mounted upon its frame. No tractor unit manufacturer can boast the complete stability of his equipment until this test has been satisfactorily completed.

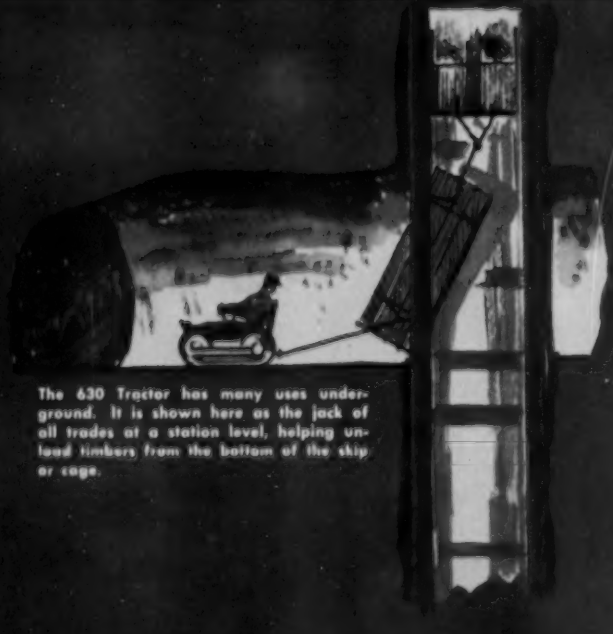
Eimco 630 tractors with excavator attachments have proved their efficiency and dependability in operation in many countries, under the most difficult conditions. Some operators have taken out track to convert to trackless operation. One customer has 16 machines installed or on order. Some contractors have completely changed over to 630's for their loading operation. One mining man says "it's the greatest thing in mining equipment since the rotating drill."

For pure enthusiasm in the field, Eimco finds its customers the best sales people. Operators, all over the world who have used the Eimco 630 can tell you the costs, the production, the increase over previous months with other equipment and their names are available to bona fide mining people.

Model 630's are built of alloy cast steel throughout. They are equipped with heavy-duty parts just as all other Eimco equipment is designed and built. Every care has been exercised to assure the customer of long life in places that have been tra-

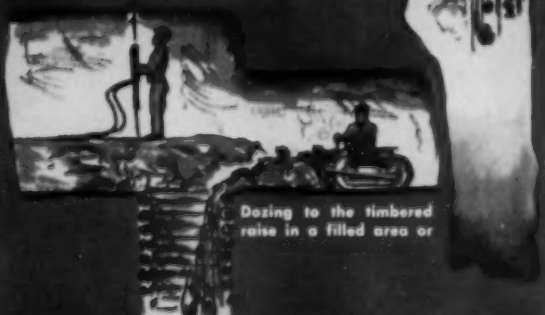


EIMCO 630 SE



The 630 Tractor has many uses underground. It is shown here as the jack of all trades at a station level, helping unload timbers from the bottom of the skip or cage.

The 630 Bulldozer does a better job of spreading fill in worked out area or



Dozing to the timbered raise in a filled area or

Moving ore in a benches have b necting man wa

ditionally weak on ordinary crawler equipment. Such items as track frames, idlers, rollers, sprockets and track chains have been given special attention in design and production.

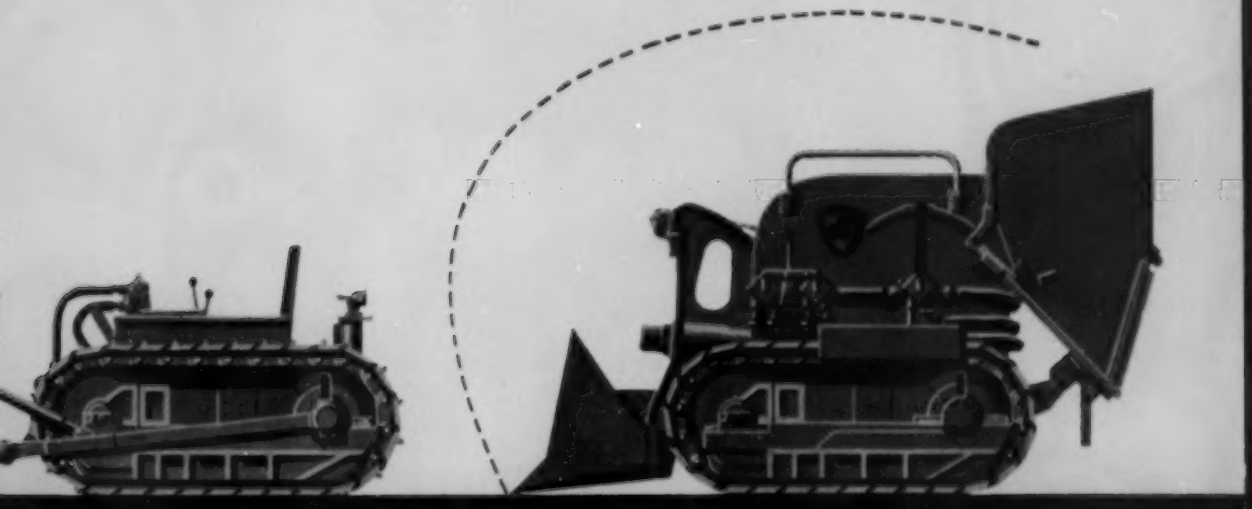
Special heat treatment is another exclusive feature of Eimco 630 parts. Each part is given the type of heat treatment most suited to producing the end result of long life in continuous operation.

All of these factors result in a machine that will work longer and harder with lower maintenance costs and a minimum of attention. Here are some of the jobs the 630 can do for you.

The 630 as an excavator will dig in and load rock, sand, gravel, conglomerate, clay or other material at an unprecedented rate. It is built in several discharge heights (see specs) to provide for low head room or high head room and under any conditions to load the car or truck full. The excavator is an ideal unit for development, drawpoint loading and production from room systems.

For extremely low head room, specify the 635 loader which meets the overall head room requirements that are under five feet. This model incorporates many new features in loading equipment and is covered by patents and patents pending as are the other 630 models. The advantages of the 635 are in fast loading and the ability to store the biggest part of a car load on its conveyor. It also provides for complete loading of long low cars in restricted head room areas.

In areas where a short haul is required, it is many times desirable to use the 630 tractor-hopper

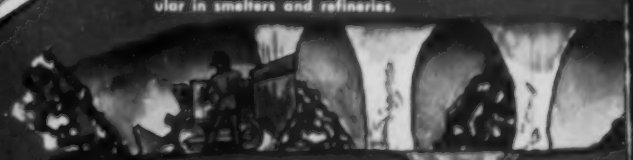


ERIES FOR LOW

ore in a shrinkage stope where have been developed by con- man ways across the stope.

The 630 loader is a good slope loader operating on grades up to 30° with no loss of traction.

The 630 Transport loader — effective for short hauls. Loads 30 cu. ft. into hopper which is integral part of machine. Discharge is automatic trip arrangement. This machine is popular in smelters and refineries.



The 630 loader is an ideal loading production tool. It loads rubber tired trucks or trains of cars.

The Hi-discharge 630 loader will load big haulage units.

630 Tractor

630 Tractor-Excavator

630 Bulldozer

WHAT IS THIS NEW ERA?

This is the era of high productivity — of mining low grade ore at a profit through the use of new mining tools and equipment.

Eimco's contribution to this new era is its new crawler Model 630 Series machine. This is not an experiment, but a thoroughly field tested unit that is meeting with outstanding success and acceptance. It gives the miner the equipment he needs to drill more holes, to bar down quickly and in safety, to load out more rock faster, even in low headroom, to back fill, to bulldoze, to set timbers, to dig, to carry, to mine pockets, to bolt roofs, to sink shafts and a thousand other jobs — and to do all these things more easily than ever before.

This new productivity is within the reach of every mining operation. Eimco offers, in its Model 630 machines, to the company crew, the contractor or the prospector a new, efficient and effective means to combat costs and increase production.

The 630 is a heavy-duty air or electric powered tractor or prime mover. It is the chassis on which we have designed and built a host of dependable, highly useful machines and attachments, some of which are shown in the next pages. As on all of Eimco's equipment, the castings are heavy alloy steel, the construction is heavy-duty throughout. The tracks on the 630 are independently controlled. There are no clutches, no foot levers, no gear shifts — just two simple handles to move forward or reverse for a similar motion in the machine. Twist the handles for a spin turn moving one track forward and the other reverse at the same time.

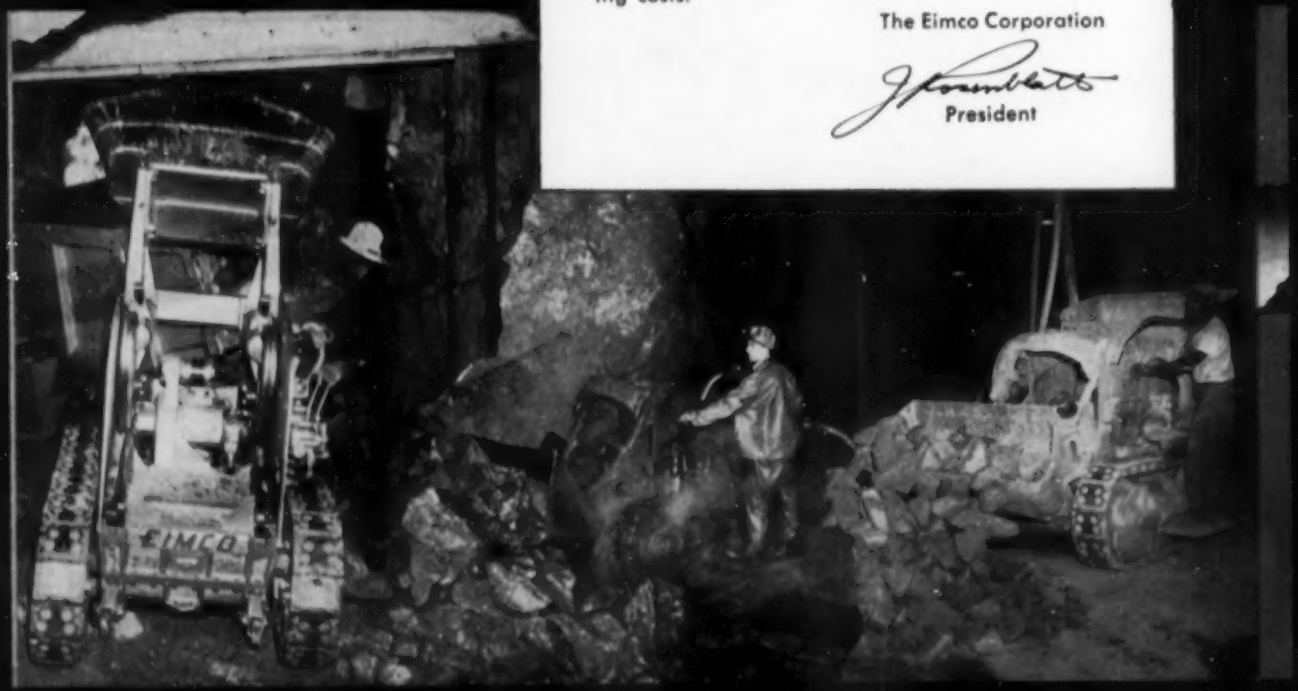
It is our belief that the 630 series of Eimco machines and equipment will make as important a contribution to trackless loading and mining as did the wheel mounted RockerShovels, which we pioneered, in their field.

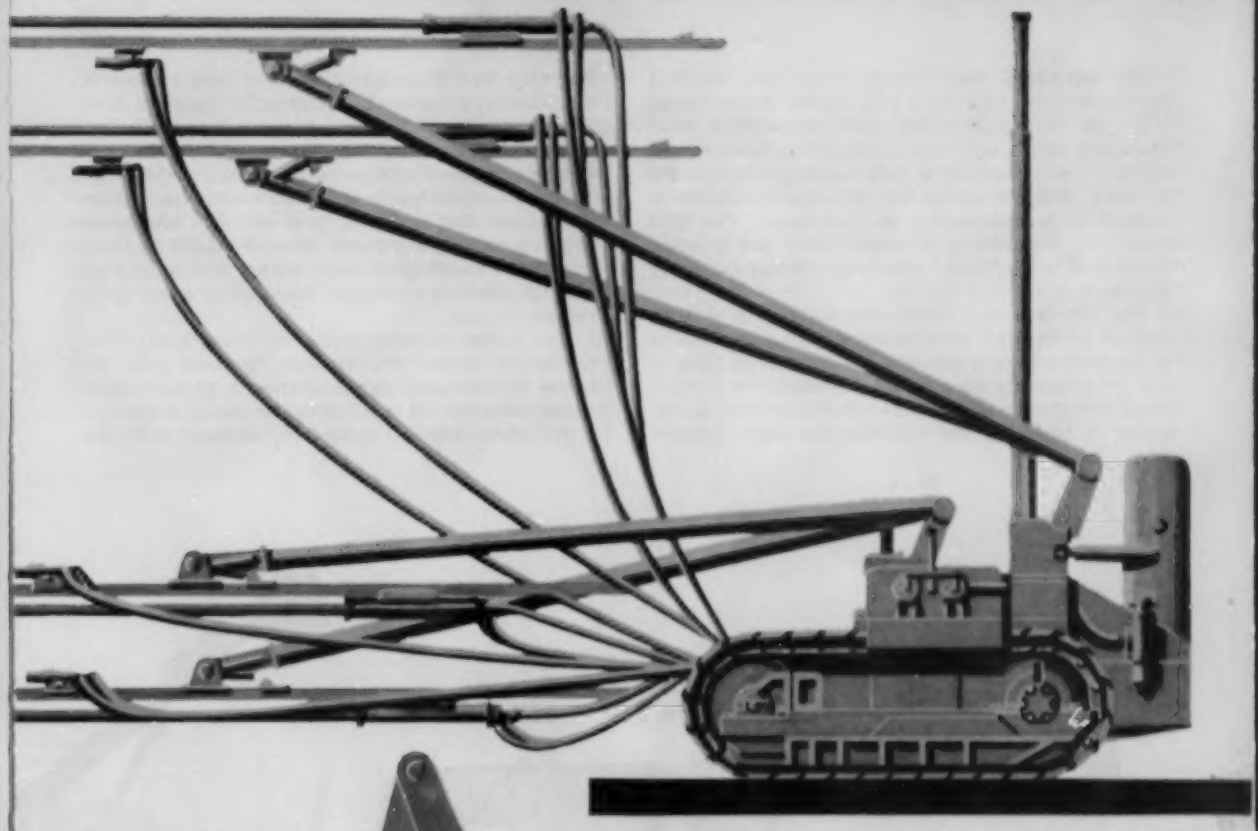
Turn the page — Eimco presents the equipment to open a new era — at an all time low in mining costs.

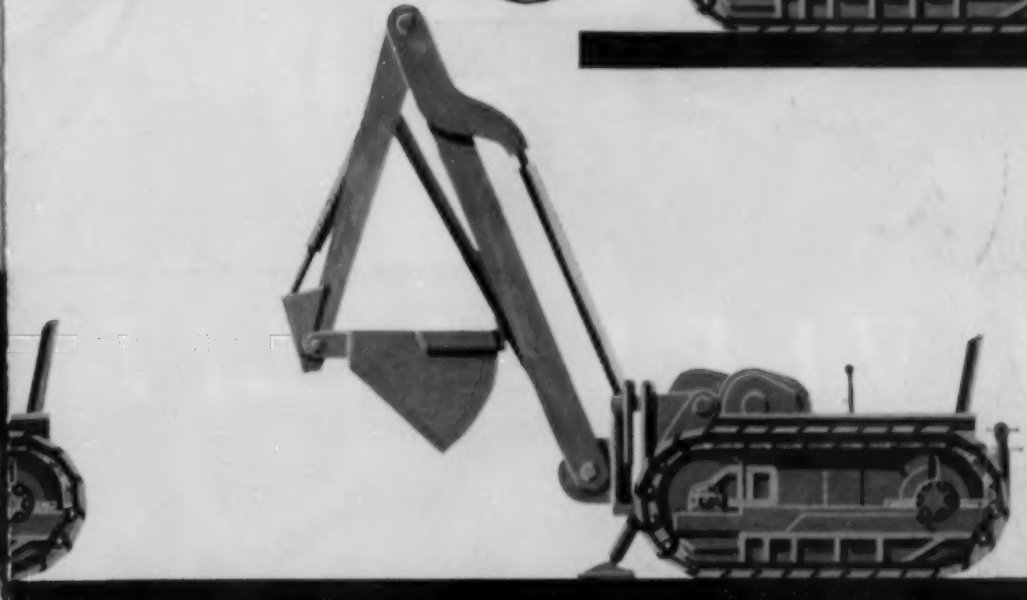
The Eimco Corporation

J. Lombard

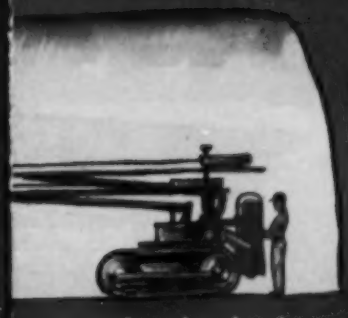
President





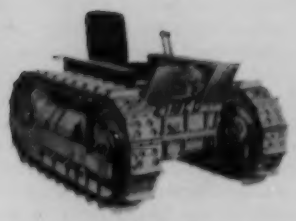


The 630 ditch digger or back-hoe has many uses on the surface as well as underground.



erators controls are located at the rear.
lds into a compact unit for moving.

The 630 Two machine jumbo. This is an easy operating unit. It can be maneuvered to drill the complete face and move out of the way for the loading equipment.



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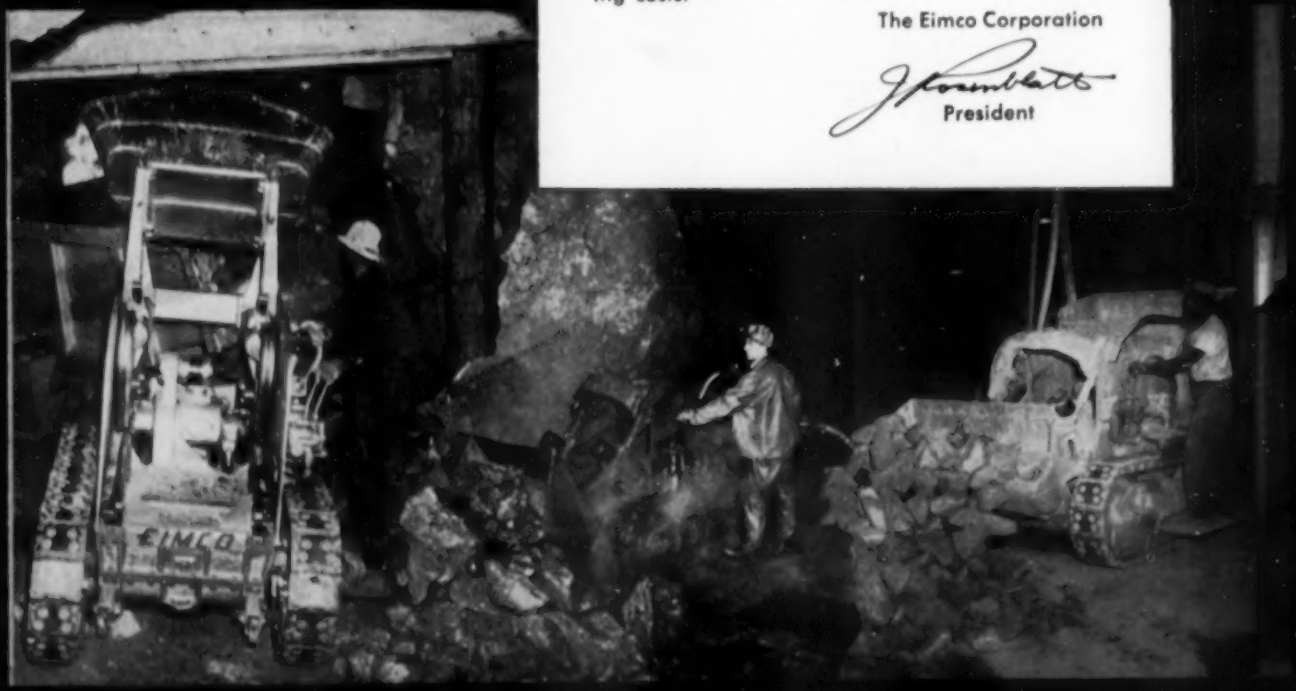
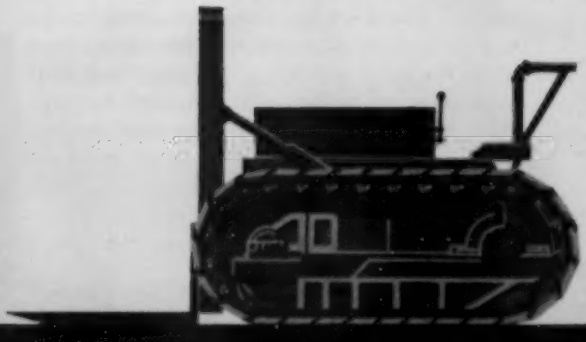


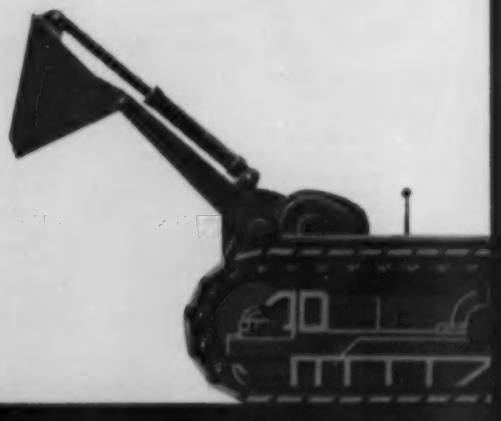
— which is an excavator unit with an integral hopper. This unit has been popular not only in mines but in smelters, chemical and pigment plants and many others where bulk material is to be moved to a hopper at floor level. The hopper on the 630 tractor-hopper can be easily adjusted to open and close automatically.

The 630 bulldozer has found an application in stopes where a drift has been cut in the ore and then widened by slicing on the wall to provide an open level for drilling vertical holes to drop the ore into drawpoints at the bottom of the stope. It has also been used for spreading fill waste in stopes and for dozing ore into the cribbed ore chute that comes up through the waste fill. Numerous other uses have been suggested for this versatile bulldozer.

The 630 tractor as a bare unit has been found better than previous methods for pulling timbers, rails and miscellaneous supplies into the station from the shaft opening. Operators claim better control and less lost time using the 630 tractor.

One important use for the 630 tractor-excavator is in shaft sinking. One operator in Virginia and one in Wyoming have used a new shaft sinking technique to employ the 630 and have reported outstanding success. These contractors use less men in the bottom and practically no set up time. Both have been impressed with the idea that when the shaft is complete they have a unit that can be used in development or production in regular mine operation and do not have the expense of single purpose shaft mucking equipment.



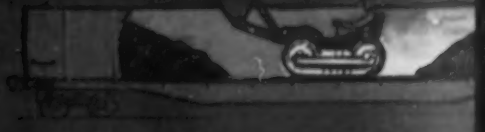


COST MINING W

The 630 fork lift has many uses underground and on the surface. Loading and unloading trains. Carrying supplies and equipment are among its many uses.

The fork lift equipped with a platform provides a substantial and easily mobile scaling platform.

The front end 630 loader is desirable for low headro and its excellent maneuverability enables peak capacity inside the confined area of a gondola or



Where room is lim has been dropped close to the haulage

Where a slice has been made and cars cannot conveniently be brought into the room.

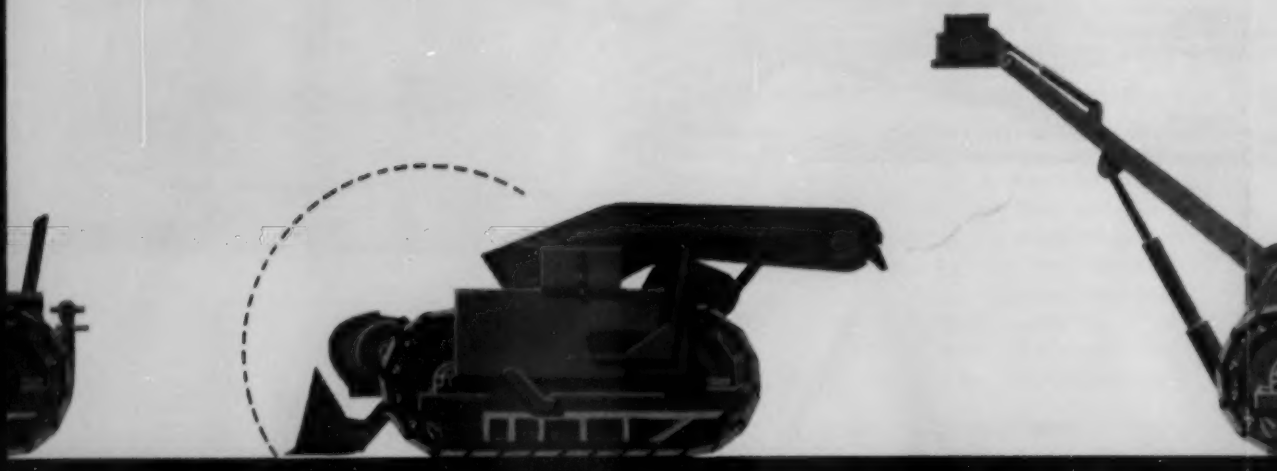
For operators who prefer front end loading equipment, the 630 front end loader is available. This type of loader can be used in restricted head room and has its application in narrow headings or rooms where loading is into haulage equipment in front of the machine or for cleanup on muck pushed to a location by the bulldozer. The 630 front end loader can be used as a combination dozer and loader and has a place in spreading or placing fill.

The 630 tractor is well adapted to the drilling portion of the cycle as a base for the jumbo. When a single machine is mounted with a short feed it can be employed as a "crust breaker." This operation is typical around pots in the aluminum industry or in breaking out consolidated slag in reverberatory type furnaces. This same type of unit has an ideal application in "scaling." Here the operator can control the drill direction accurately and break down loose material from any location in the heading because of the maneuverability of the 630.

Forklift attachments are used in the stations underground for removing and piling timbers and lagging from the train at desired locations, carrying to the headings or raise areas. These units can also be used as platforms for driving lagging between the sets.

The timber setter is used for lifting and setting the heavy timber cap in heavy ground. This unit saves time in getting the timber in place and reduces time lost in production or development.

For mines using "roof-bolting" instead of timber,



ITH VERSATILITY

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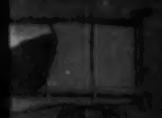
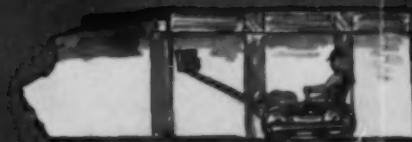
loading, even



ted and muck
from a raise
way or

The ultimate in low headroom loading equipment is the 635 Loader. This versatile machine provides high speed loading where seams are thin and a minimum of excavation is desirable.

The 635 Eimco Loader is ideal for long low cars and the "shuttle-buggy" type of haulage equipment.



The 630 timber setter is another version of this v
Eimco air powered tractor. With independent tra
trol, the 630 provides the only mobile unit th
place timber without numerous movements to "li

The 630 roofbolter. Another unit unsurpassed in speedy
ation because of the maneuverability of the Eimco tractor
Holes can be located quickly and efficiently.

the 630 tractor provides an excellent mounting for roof bolting apparatus. The easy operation and extreme mobility of the 630 makes it ideal for this type of work.

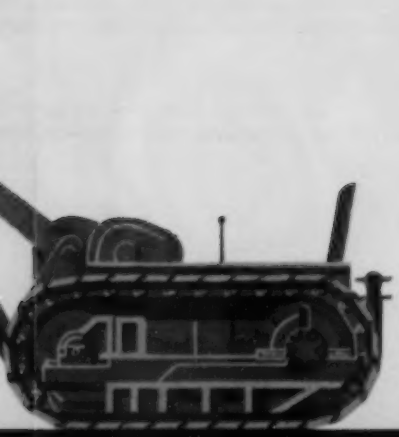
Where comparatively low head room requires the mobility of low equipment, the 630 tractor can be equipped with drill mountings and used as a jumbo.

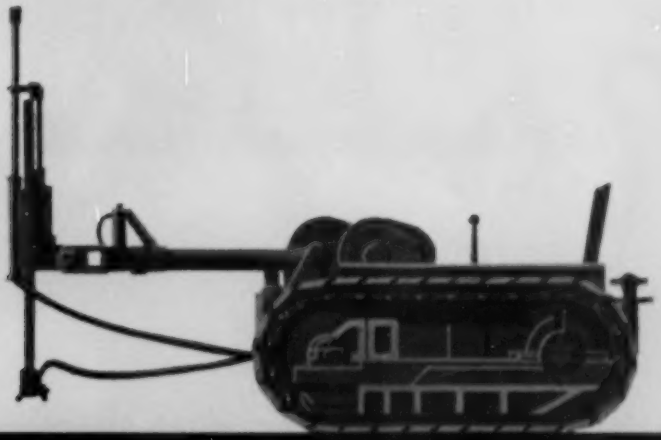
Two drill jumbos for the average heading, positioned, raised, lowered or moved at any angle give new freedom to the miner and speed up the drilling program.

High speed tunnels and production jobs can use the long base 630 tractor for mounting as many as four drill rigs with its separate controls for all movements and operated from the rear of the tractor. The four machine unit will drill out a round quickly and provide for maximum number of rounds per shift.

For drainage ditches, sumps, and other excavation below grade and in confined spaces, the 630 can be equipped with a back hoe with 180° swing for loading into cars or rubber tired haulage equipment.

The 630 tractor-excavator combination is excellent for loading on slopes. Grades to 30% require no cables or other anchoring devices to operate efficiently. It is suggested, however, that if the 630 tractor-excavator is being considered for slope operation that such information be given to the Eimco engineer so that the step plate can be modified for the proper angle and the bucket be adjusted to the proper discharge angle.





Y UNDERGROUND

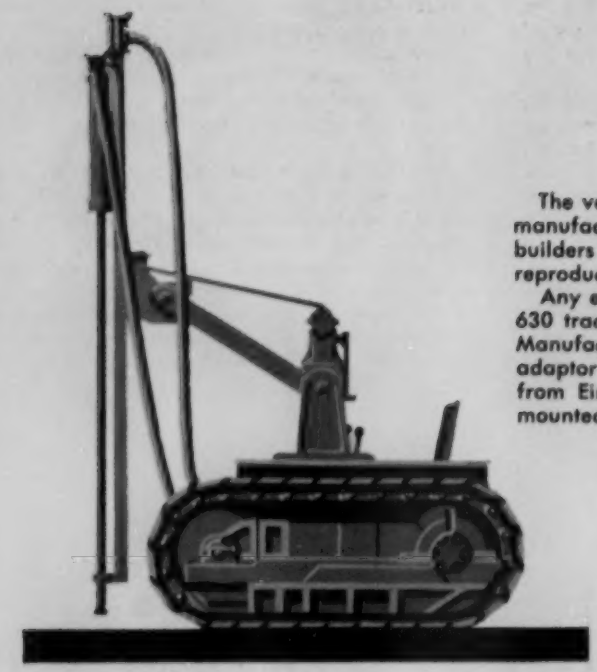
of this versatile
ndent track con-
unit that will
nts to "line-up."

speedy oper-
tractor unit.

The 630 "crust-buster" designed to break out scale or take up bottom in a reverberatory type furnace.

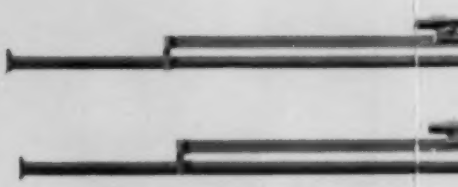
Another adaptation of the "crust-buster" is the 630 "scaler." This gives the miner an opportunity to scale the entire face quickly and efficiently, while sitting down in a safe position.

The 630 Eimco loader as a shaft mucker. This machine has been used successfully in several shafts and works effectively in either round or rectangular shafts. The 630 loads faster, requires no special set-up, uses less men than specialized equipment and provides a standard type drifting machine when drifting starts at the desired level.

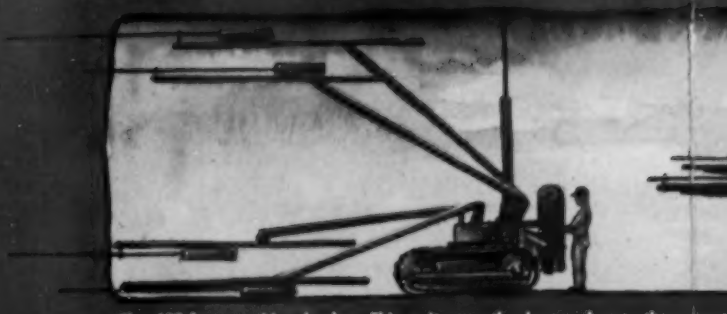


The various attachments shown on these pages are manufactured by numerous specialized equipment builders and no particular brand is intended to be reproduced in these sketches.

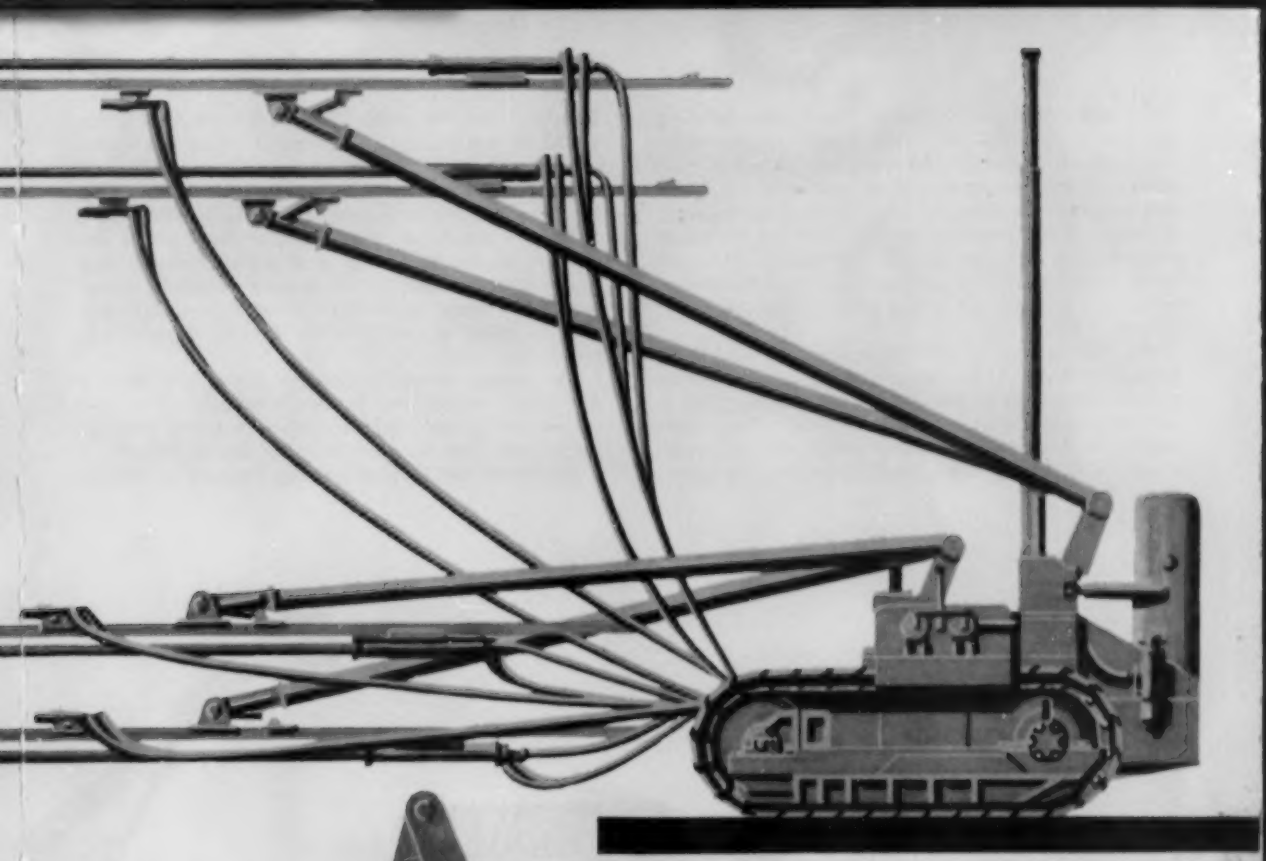
Any equipment suitable for mounting on the Eimco 630 tractor base can be shipped directly to the job. Manufacturers of the various attachments can provide adaptor plates or secure standard drilling patterns from Eimco so that their attachment will be easily mounted.

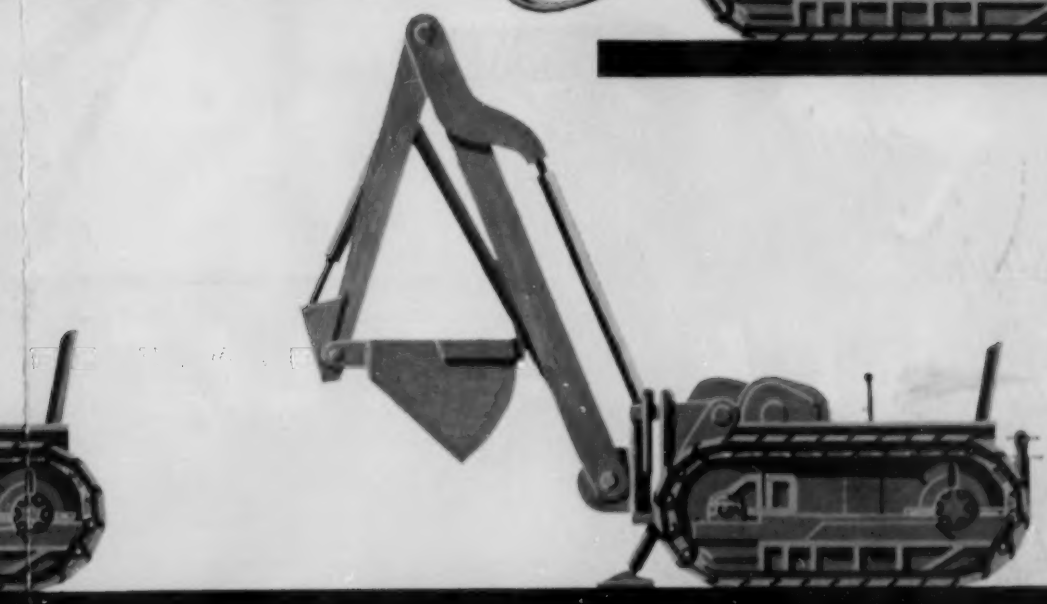


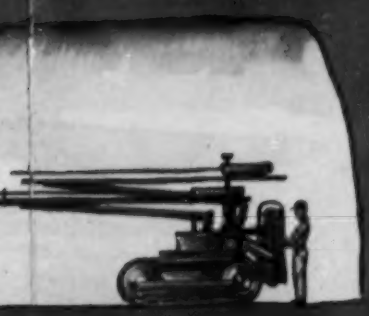
The 630 tractor drill replaces the old style wagon drill. The operator spots the hole, drills the hole and drives the rig from the sitting position. He can drill more holes per shift with less effort.



The 630-four machine jumbo. This unit uses the long base Eimco tractor and because of the numerous controls used in raising, lowering, moving and drilling with each boom, the operat
The jumbo folds







e operators controls are located at the rear.
o folds into a compact unit for moving.

The 630 ditch digger or back-hoe has many uses on the surface as well as underground.

The 630 Two machine jumbo. This is an easy operating unit. It can be maneuvered to drill the complete face and move out of the way for the loading equipment.

EIMCO 630 SPECIFICATIONS

630 TRACTOR

	ENGLISH	METRIC
Overall Height	45"	1145mm
Track Height	28"	710mm
Grouser Height	1 1/8"	30mm
Overall Width	55"	1395mm
Track Shoe Width (standard)	9"	230mm
Track Gauge	45"	1145mm
Ground Clearance	6"	150mm
Oscillation at Front Idler	6"	150mm
Overall Length	71 3/8"	1825mm
Length of Track on Ground	43 7/8"	1115mm
Height of Drawbar Above Ground	9 5/8"	245mm
Drawbar Tongue Opening	2"	50mm
Drawbar Pin (diameter)	1 1/2"	40mm
Air Inlet — Pipe Size	2" IPS	
Weight Complete	6500 lbs.	2950 kg.
Area of Ground Contact (9" trackshoe)	790 sq. in.	5100 sq. cm.
Speed	0-1.5 MPH	0-2.4 km./hr.
Motors (Eimco Air or A.C. Electric)	3	3
Electric Motor Rating	SPECIAL	
Air Motor Rating	12 HP @ 90 psi	
Drawbar Pull (lbs.) at Zero Track Slippage	10600 lbs.	4800 kg.

630 LONG FRAME TRACTOR

Overall Length	88"	2235mm

630 TRACTOR-DOZER

Overall Length	91 1/8"	2320mm
Blade Pitch Angle Adjustment	10°	18°
Maximum Blade Lift	16"	405mm
Maximum Blade Dig	7"	180mm
Maximum Blade Tilt	8"	205mm
Blade Height	26 3/4"	680mm
Blade and Overall Width (Longer Blades Available)	62"	1575mm
Weight Complete	7980 lbs.	3620 kg.

630 TRACTOR-EXCAVATOR

Overall Operating Width	5'8 3/4"	1745mm
Minimum Height	4'11 1/2"	1510mm
Overall Length (bucket down, std. bucket)	9'4"	2845mm
Minimum Length	6'3 1/2"	1910mm
Maximum Digging Depth Below Grade	22"	560mm
Head Room	6'8"-10'7"	2030-3225mm
Discharge Height	3'9"- 7'2"	1145-2185mm
Bucket Capacity	5 1/2-11 1/4 cu. ft.	1/6-1/3 cu. meters
Weight Complete	9400 lbs.	4260 kg.

630 TRACTOR-HOPPER

Width of Hopper	4'6 1/2"	1385mm
Overall Length (bucket down)	10'6"	3200mm
Length of Tractor (with hopper)	8'0"	2440mm
Hopper Capacity	32 cu. ft.	0.9 cu meters
Weight Complete	12000 lbs.	5445 kg.

635 TRACTOR-LOADER

Overall Length (bucket down)	10'9"	3275mm
Minimum Headroom	5'0"	1525mm
Bucket Capacity	5 cu. ft.	1/6 cu. meter
Belt and Hopper Capacity	15 cu. ft.	1/2 cu. meter
Weight Complete	11000 lbs.	4990 kg.

The Eimco 630 Tractor is an air or electric powered prime mover that can be fitted with a special excavating attachment and other tools.

This rugged, dependable machine is the answer to the needs of miners and contractors alike. The maximum of trouble-free operation and long-life has been built into each and every part of the Eimco 630 Tractor.

Many new features and advantages in tractor design have been incorporated in the 630. Independent track control provides a pivoted turn by running one track forward and the other track reverse. This feature makes it possible to maneuver from any angle without backing to make a new approach. All wearing parts are easily replaceable in the Eimco design. All parts are easily accessible for quick, easy maintenance work. Tracks oscillate for operation on uneven ground. Tracks are equipped with spring release to clear rocks or obstructions in the tracks. Front idlers are solid web type, one piece cast steel. Track links are heat-treated alloy-steel forgings. Track pads are heat-treated alloy-steel castings, with integral cast grousers if required. Rigid cast steel diagonal brace construction provides track alignment that stays true. Heavy shafts and oversized anti-friction bearings are used throughout.

Controls are simple. Two handles, conveniently located in front of the operator, effect all tractor movement.

Mounting pads are provided with suitable tapped holes for mounting many types of tractor attachments.

All parts are carefully heat-treated for maximum service and minimum wear. All parts are of heavy alloy-steel cast or fabricated construction for typical Eimco long-life and trouble-free heavy duty operation.

Write or call the nearest Eimco Office for more information on the modern Eimco 630 Tractor.

THE EIMCO CORPORATION

EXECUTIVE OFFICES AND FACTORIES: SALT LAKE CITY 10, UTAH
EXPORT OFFICES: EIMCO BLDG., 52 SOUTH ST., NEW YORK CITY

BRANCH SALES AND SERVICE OFFICES: New York, N. Y., 51-52 South St. — Birmingham, Ala., 3140 Fayette Ave. — El Paso, Texas, Mills Bldg. — Chicago, Ill., 301 So. Hicks Rd., Palatine — Duluth, Minn, 216 E. Superior St. — Kellogg, Idaho, 307 Division St. — San Francisco, Calif., 637 Cedar St., Berkeley — London W.1, England, 190 Piccadilly — Houston, Texas, 4008 Purdue St. — Pasadena, Calif., 434 No. Lake Ave. — Baltimore, Md., P. O. Box 1052 — Pittsburgh, Pa., Investment Bldg. — Seattle, Wash., 208 Hoge Bldg.

AFFILIATED EIMCO COMPANIES: Societe Eimco, 29 Rue De Mogador, Paris 9, France — Eimco (Great Britain) Ltd., Gateshead-on-Tyne 11, Co. Durham, England — Eimco Italia, S.P.A., Via Senato 11, Milan, Italy — Eimco (South Africa) Pty. Ltd., 136 Kinden Road, Robertsham, Johannesburg, South Africa — Eimco Canada Limited, 320 Industrial Ave., Vancouver 4, B. C., Canada.

B-193

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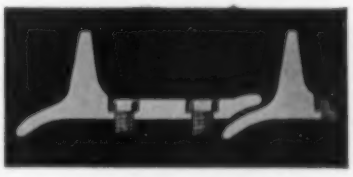
Switch to extra-tough AMSCO® MANGANESE STEEL TRACTOR SHOES

When the going is extra tough, as in rocky areas or abrasive mineral soils . . . tractor shoe replacement can become a major cost item. Both repair time and down time eat up profits.

Switch to "the toughest steel known" . . . Amsco Manganese Steel . . . for tracks and grouser bars. Check their much longer service life against the moderate extra cost. Add to this the greater efficiency and pulling power of your tractor, over a longer period of time. The answer: *important operating economies.*

Amsco Manganese Steel gives excellent resistance to abrasion accompanied by impact . . . actually *work-hardens* in use. Write for full information on long-wearing Amsco Manganese Steel Tractor Shoes—the economical answer to high track-replacement costs.



COUNTER-SUNK BOLT HOLES

Amsco Track Shoes have holes countersunk for less wear on bolts. Saves cost of replacing bolts when changing shoes.

AMERICAN Brake Shoe COMPANY

AMSCO

American Manganese Steel Division • Chicago Heights, Ill.

OTHER PLANTS IN: DENVER, LOS ANGELES, NEW CASTLE, DELAWARE, OAKLAND, ST. LOUIS, JOLIETTE, QUEBEC

GISMO METHOD SETS NEW

New Production Records and Drastic Cuts in Man-hour Costs Foreseen!

COULD THIS BE THE BEGINNING of the "Gismo Age" in rock mining? One mine operator has said: "The Gismo is going to revolutionize rock excavation methods in much the same manner that large tractor drawn earth moving scrapers have revolutionized the earth moving procedure."

One new mine now being mechanized by the Gismo Method contemplates a production of 2,000 tons per day at an overall cost of about $1.00 per ton. Production is forecasted to be approximately 50 tons per man when operated on a 2,000 ton per day basis and 57 tons per man on a 4,000 ton basis. The deviser includes all men to be employed for production, surface and underground, including all supervisors. The above projection is based on results obtained over a three year period by this system in a similar mine.

An older mine, changing over to the Gismo Method, has increased tonnage per man-shift almost 10 times in the stoping operations. Compared to the conventional method used in 1950, the Gismo Method has halved both development and labor costs.

What is this equipment? It is a general purpose utility self-loading transport that is simple, versatile and built to take the punishment of rock excavation. No special development facilities or conditions are usually required. It operates in sloping ore bodies with irregular outlines, as well as large or relatively small openings. It loads in development or production . . . transports . . . supports 2 to 5 jib mounted drills . . . back fills, moves boulder rocks . . . makes its own roadways and cleans up completely. It can load and transport up to 100 tons per hour with a 300 ft. haul.

It's an extremely simple piece of equipment requiring little maintenance. The Gismo's initial cost can be reasonably compared with one year's maintenance cost alone of much of the conventional equipment in present use.

It makes possible a mining method requiring few machines, few men, less planning and integration. It permits total mechanization of your mine—the solution to high man-hour and materials cost.

We urge you to compare the Gismo Method with other methods—from initial investment to overall effect on your operation's efficiency. Call, wire or write us today! *Sanford-Day Iron Works, Inc., P. O. Box 1511 . . . Telephone 3-4191, Knoxville, Tennessee.*

Sanford-Day IRON WORKS

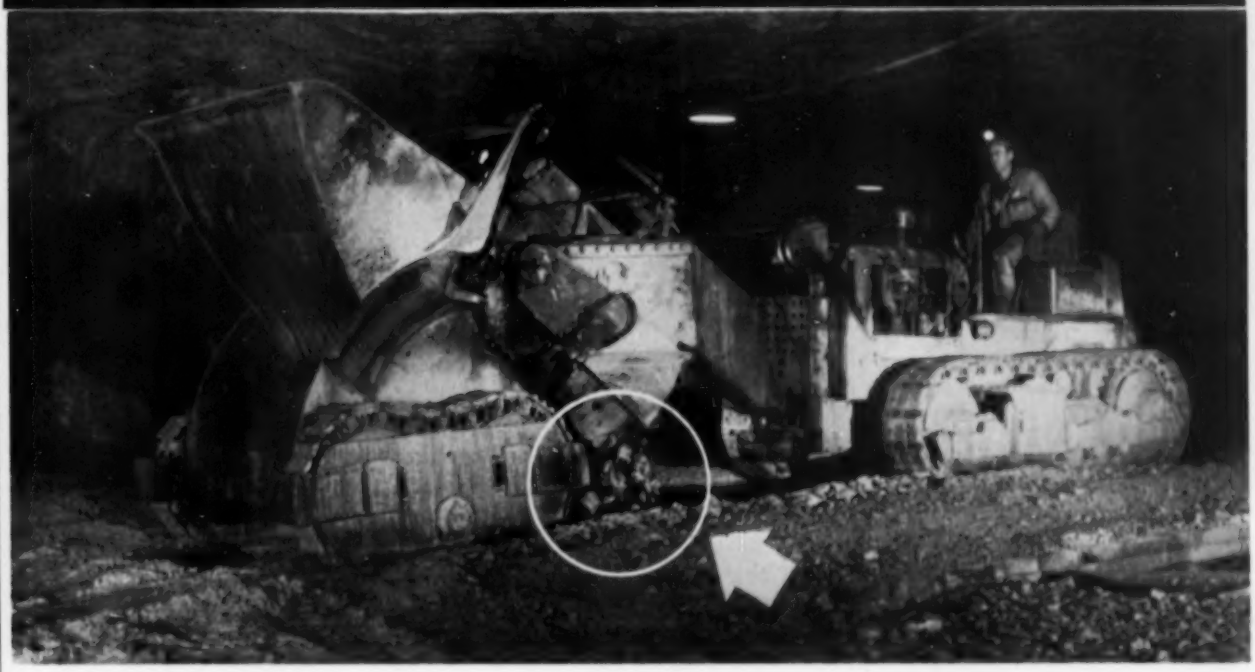
KNOXVILLE • TENNESSEE

AND BROWN-FAYRO DIVISION

MINE CARS, All Types - PRECISION WHEELS - "Brownie" HOISTS
CAR RETARDERS - SPOTTERS
PUMPS - OIL SPRAY SYSTEMS

LOADING (MUCKING)— a one-man operation! The speed of the GISMO Self-Loading Transport can be fully appreciated and understood only when seen in actual operation. With sufficient ore and ample surge capacity, one GISMO as a self-loading transport and one GISMO equipped as a stope drilling jumbo will excavate out of a hard rock face over 400 tons in a six hour working time shift. The GISMO handles boulders . . . sets them aside for secondary blasting . . . and does not require a bulldozer, road grader or any other accessory device. The operator has complete control of his shoveling grade—can always look directly at the dipper edge. The shoveling action is gentle. Carrying capacity of the GISMO shown working above is 6 tons. Other capacities are available.

GOALS FOR ROCK MINING!



DUMPING: direct into surge pocket . . . skip pocket . . . conveyor for truck loading . . . or mine cars with ramps (for development work). GISMO above is dumping load into surge pocket. Both when mucking (loading material into body of GISMO) and when dumping the load, the dipper or shovel head is in position shown above. The surge pocket disconnects from a time cycle standpoint the next operation, transportation—permitting GISMO and the transportation system to operate independently of each other for maximum efficiency and production.

TRAMMING: fast and highly maneuverable! GISMO shown above is transporting load to a surge pocket. When emptied, GISMO will return to reload with dipper in position shown. As you can guess, the GISMO is a very simple machine functionally and mechanically. Compared to any other loading or mucking device (none of which are transports), the GISMO has a very large capacity.

DRILLING: The GISMO will mount 2, 3, 4 or 5 drills at ideal locations for the efficient drilling of small or large headings, benches, rib-slabbing or back-slabbing. The GISMO Stope Drilling Jumbo mounting four drills and operated by two men often averages by the month over 400 feet per man shift. The Gismo Drill Jumbo can be easily converted to a Gismo Self-Loading Transport.

IMPOSSIBLE WITHOUT EXPLOSIVES

The modern highway—wide and spacious with increased visibility and no cross traffic—is a tribute to modern construction techniques. The above section of U.S. 99 between Bakersfield and Los Angeles, California, is a good example of engineering know-how. Where rock was encountered, blasting crews with Hercules explosives took over.

For more than forty years, Hercules has engaged in continuous research into the development of explosives materials and improved blasting techniques. Whether your requirements are for construction, mining, quarrying, seismic prospecting, or in other fields where explosives are needed, Hercules technical representatives will be glad to assist in the selection of the right materials for the most efficient job.

HERCULES POWDER COMPANY

Explosives Department, 955 King Street, Wilmington 99, Del.

Birmingham, Ala.; Chicago, Ill.; Duluth, Minn.; Hazleton, Pa.; Joplin, Mo.; Los Angeles, Calif.; New York, N. Y.; Pittsburgh, Pa.; Salt Lake City, Utah; San Francisco, Calif.

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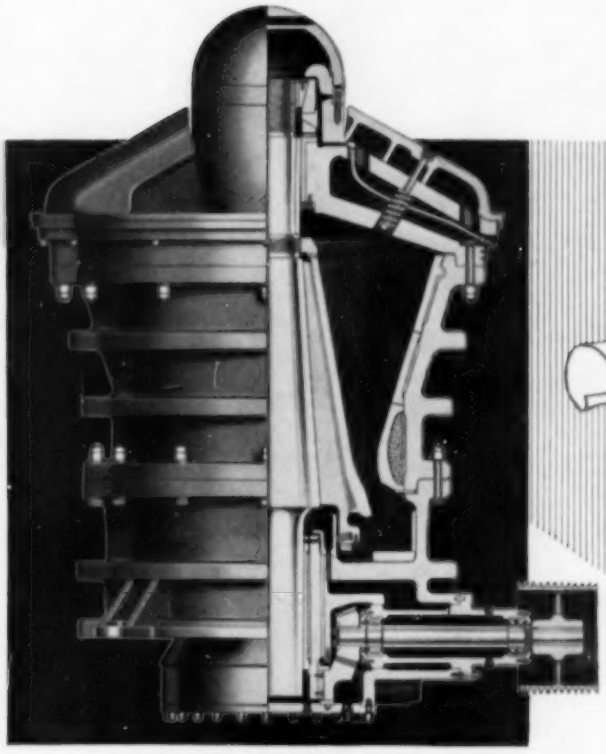


The One Crusher That Is

COMPLETELY ADAPTABLE

to Changing Operating Conditions

YOU GET MORE when you specify Allis-Chalmers — more tonnage, more operating economy. Only the *Superior* crusher gives you the built-in flexibility that promotes maximum efficiency from your *entire circuit* at all times.

Capacity demands can be met by changing countershaft speed and eccentric throw. Product size control is obtained by modifying shape of crusher chamber and by vertical adjustment of the mainshaft.

If your *Superior* crusher is equipped with *Hydroset* vertical adjustment (optional), changing crusher setting to compensate for wear or to vary product size is a one-man, one-minute operation. *Hydroset* control also facilitates emptying crusher chamber in case of power failure or other emergency.

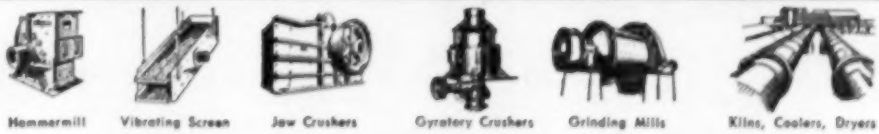
For complete information, see your A-C representative or write Allis-Chalmers, Industrial Equipment Division, Milwaukee 1, Wisconsin, for Bulletin 07B7870. A-4989

Primary and Secondary

GYRATORY CRUSHER

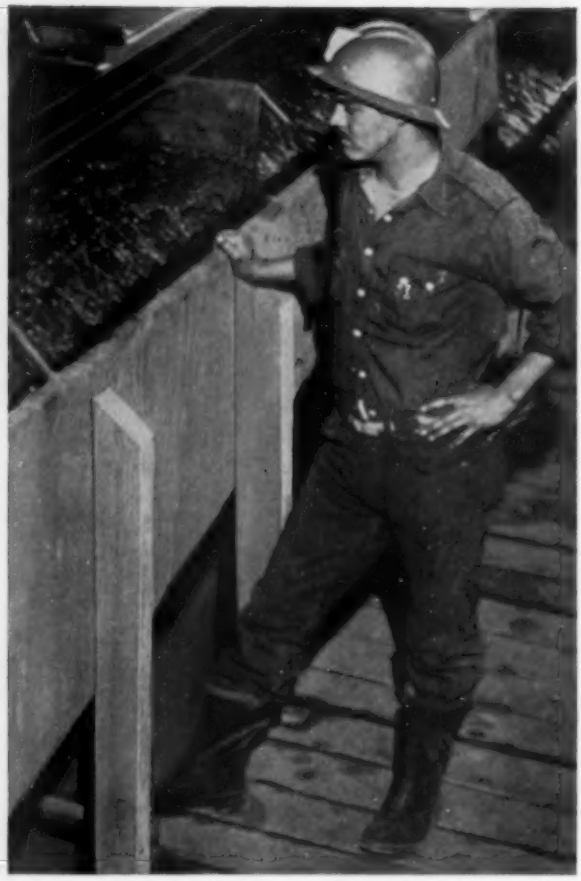
Superior and *Hydroset* are Allis-Chalmers trademarks.

ALLIS-CHALMERS



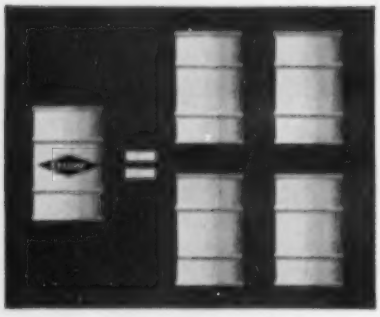
Hammermill | Vibrating Screen | Jaw Crushers | Gyratory Crushers | Grinding Mills | Kilns, Coolers, Dryers

DOW



Lab and mill men agree—Dowfroth is effective with as little as one-fourth the quantity of other frothers. The metallurgical results are excellent.

point of agreement: Dowfroth 250 most effective



Money-saving arithmetic. One drum of Dowfroth 250 can equal four of other frothers on your ore. Dowfroth 250 also has definite uniformity of properties and is readily available in drum or tank-car quantities.

Lab metallurgist or flotation operator—both prefer DOWFROTH!

The man in the lab—because Dowfroth® 250 is noncollecting, is water soluble and produces a clean, lively froth over a wide range of conditions. It gives him more precise control, yet better accommodation to test variations.

The man in the mill—because the action of Dowfroth 250 is rapid and under control—because Dowfroth gives concentrate froths that are easy to launder, pump and thicken—because Dowfroth doesn't deteriorate rubber parts—is pleasant to handle.

There's still more to the story. Management prefers Dowfroth 250 because it makes milling more profitable. Superior water solubility means more frother returned in water recovery systems. Dowfroth improves concentrate grade and mineral recovery.

Save money in your flotation *process* by using Dowfroth 250—and the superior collecting agents, Dow Xanthates. A new Dow product is Separan 2610®, a superior flocculant already popular in the industry. Samples of these products and technical assistance are offered by THE DOW CHEMICAL COMPANY, Midland, Mich., Department OC 856M.

you can depend on DOW CHEMICALS DOW

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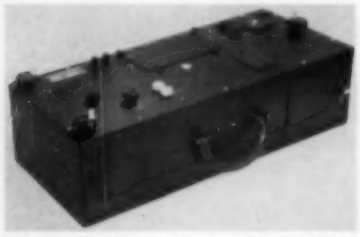
Portable Seismograph

W. F. Sprengnether Instrument Co. has a portable seismograph for recording blast vibrations. Readying for use requires only several min-



utes. Unit is self-contained in a rigid aluminum case, 25x10x8 in., weighs 38 lb, and requires no external connections. Manufacturer will arrange for the analysis of recordings if desired. **Circle No. 1.**

Cutter Bits

The U-10 bits by the Mining Tool Div. of *Kennametal Inc.* are reported to produce coarser cuttings at lower power cost. Originally designed for use in potash mining, they have also been proven in salt and coal operations. High wear resistance makes stoppages for replacement or spotting less frequent. **Circle No. 2.**

Twin-Diesel Tractor

LeTourneau Westinghouse Co. Twin-C tractor is claimed the world's most powerful. Its synchronized twin diesels deliver a total of 416 hp and

the 40-ton tractor moves at speeds up to 20 mph. It is applicable to operations otherwise requiring tandem pushers. Six-foot high tires take advantage of weight and are hydro-inflated. **Circle No. 3.**

Compressor-Rock Drill Team

Gardner-Denver Co. has heavy-duty equipment for contractors, rock quarries and open pit mines. Their Rotary 900 compressor delivers 900 cu ft of air per min and is of simple design, permitting easy inspection and maintenance. The DH143 is a percussion drill with a 5½ in. hammer for fast penetration of hard rock. A selection of bit sizes from 3½ to 5 in. is available to give best spacing and fragmentation efficiency. **Circle No. 4.**

Hydraulic Back-Hoe

The Sherman Major is a power digger attachment for Fordson-Major tractors providing fast, efficient digging through any soil. The unit, by *Sherman Products Inc.*, has a 12½ ft

depth reach, a 180° arc of swing and a bucket lift height of over 8 ft for loading trucks. **Circle No. 5.**

Prospecting Drill

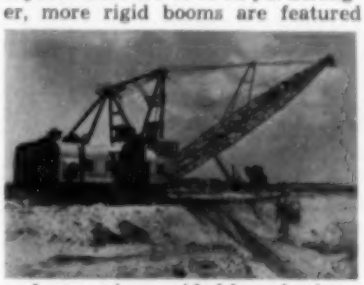
Hossfeld Mfg. Co. drill is powered by a 5-hp, 4-cy gasoline engine. Screw-on detachable rock bits in standard 1-in. thread and stock sizes from 1¾ to 2⅝ in. are used. Unit weighs 1300 lb with counterweights and can be knocked down to 100-lb sections in minutes. Using water to slush cuttings, it drills well in hard and soft formations, only exception being rock of granite hardness. **Circle No. 6.**

1850°F Furnace

Hevi Duty Electric Co. has a large size automatically controlled muffle furnace for continuous operation to 1850°F. Self-contained unit has all controls and instruments necessary for operation in furnace base. Heating chamber is 11x14x8 in. Counterbalanced hinged door swings upward to keep hot side facing away from operator. **Circle No. 7.**

Walking Draglines

Page Engineering Co. has a line of single deck walking draglines in capacities from 5 to 15 cu yd. Stronger, more rigid booms are featured



and power is provided by a horizontal V-type diesel engine of new design. Model 721 shown above has a 125-ft boom and a 7-cu yd bucket. **Circle No. 8.**

Washer-Screener

Pioneer Engineering Works has a portable sand and gravel washing and screening plant designed to meet various needs and roadside production in particular. Since local conditions vary greatly, only the basic unit is offered as standard equipment. Optional units to suit special needs are available. The basic unit consists of a three-deck gradation screen, a double-screw fines material dehydrator, an on-the-plant feeder conveyor, and two side delivery conveyors. The plant turns out two sizes and oversize of gravel and one size of washed sand simultaneously. **Circle No. 9.**

Hand-Portable Generator

Model 5903 by LearCal Div. of *Lear Inc.* is a lightweight auxiliary unit supplying either 1350 w of d-c power or 1500 w of 400 cycle a-c

power. Gas-driven plant is air-cooled, has an automatic rewind starter, and is equipped with a muffler. It weighs less than 35 lb. **Circle No. 10.**

News and Notes

Dorr-Oliver Inc. has acquired all the business and substantially all of the assets of *Merco Centrifugal Co.*, San Francisco, in exchange for approximately 50,000 shares of Dorr-Oliver common stock. Merco Centrifugal Co. manufactures and sells centrifuges, centrifugal screens and strainers. Similar equipment is not now built or sold by Dorr-Oliver. . . . *Galigher Co.* has been appointed sales representative for the OCC Vessel Heavy Media Separator for the territory west of the Rocky Mountains and *Continental Sales & Equipment Co.* is representative for Minnesota, Wisconsin, and Upper Michigan. . . . *West Winds Inc.* of San Francisco and *Keystone Engineering Co.* of Los Angeles have been appointed as facilities for the repair and rebuilding of Symons Cone Crushers by *Nordberg Mfg. Co.*

see for yourself

WHY THE WEMCO HMS DRUM DOESN'T WEAR OUT

Operators have observed that a Wemco HMS Drum Separator has a long long life.

The difference between this separator and others (that wear in a hurry) is that the entire Wemco Drum rotates as a unit. There are no moving parts inside – no chains, sprockets, screens, scrapers or sweeps to wear due to the extremely abrasive action of rock and medium. The lifters are a built-in part. In them the sink material stays in place until it drops out at top center. And even if the drum is shut-down fully loaded, there are no jamming or abrasion problems on start-up.

The common sense in this drum design is another example of Wemco's sound solution to mining industry problems. This Drum has become the most widely used of all HMS separatory vessels. It has helped, too, to make Wemco's HMS Mobil-Mill the world's most popular Heavy Media plant. Over 50% of all HMS plants use Wemco equipment. Write today and let us send you all the facts.

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WESTERN MACHINERY COMPANY
762-C FOLSOM STREET • SAN FRANCISCO 7, CALIF.
Representatives in principal cities of the United States and Canada and in major countries throughout the world.

(21) COAL CLEANING: A 40-page bulletin from *McNally Pittsburg Mfg. Corp.* has photographs, diagrams, and illustrations on coal cleaning processes. Cleaning advantages pointed to are upgrading, maxamum recovery, and salability of low-grade seams.

(22) DRAGSCRAPERS: *Sauerman Bros.'* catalog A covers DragScraper machines with details and specifications on standard installations, rapid shifting scrapers, track cable machines and tower excavators. Job photos and typical layouts supplement the 5-section booklet.

(23) RUBBER PRODUCTS: Among the industrial rubber products listed in a 12-page catalog by *Boston Woven Hose & Rubber Co.* are belting, packing, tape, and hose items with cutaway views and photos.

(24) ROLLER BEARINGS: A 72-page catalog on Shafer self-aligning roller bearings from *Chain Belt Co.* lists models with shaft sizes from ¾ to 7 in. Revised engineering data and new load rating tables are included.

(25) ARCHITECTURAL PRODUCTS: The Stran-Steel line of architectural products is detailed in a 24-page catalog by *Stran-Steel Corp.* New products listed are metal curtain wall panels and 10 and 12-in. joists. The steel curtains feature installation by means of metal clips fastened by a hand-operated tool.

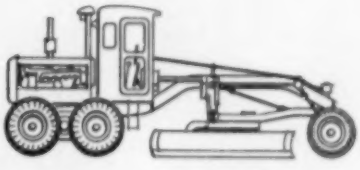
(26) MACHINING STAINLESS TUBING: Folder TDC-182 from *Babcock & Wilcox Co.* furnishes information about various machining operations on stainless pipe and tubing with pointers on tooling, power requirements, cutting oils and finishing.

(27) HOISTS AND CAGES: Bulletin KA-5601 has information on new light-duty gasoline mine shaft hoists and cages with bottom-dump skips by *Vulcan Iron Works Co.*

Free Literature

(28) RAIL CAR SWITCHERS: *LeTourneau-Westinghouse Co.* offers a bulletin entitled "SwitchMobile—diesel railroad switcher on rubber tires." The SwitchMobile claims the advantage of not following rails or roads in making operational rounds.

(29) MOTOR GRADERS: Intended as a guidebook, "When You Buy A Motor Grader" by *Caterpillar Tractor Co.* has information in non-technical language on motor grader



performance, operator convenience, operating costs. Also available in Spanish, French, and Portuguese.

(30) STEEL MATTING: Two kinds of steel floor armor to prolong floor life are available from *Klemp Metal Grating Corp.* Hexsteel is designed for ramps, docks, aisles, highway and railroad bridges as well as industrial floors. Floorsteel, a flexible matting, can be used on irregular surfaces.

(31) WATER HEATING: Wolverine Tube, division of *Calumet & Hecla Inc.* has a booklet on the use of Trufin W/H, an integrally finned tubing, in the manufacture of tankless water heater coils.

(32) MINING TOOLS: The Mining Tool Div. of *Kennametal Inc.* has new booklet M-8 on relief panel design cutter bits; rotary drill, core and drag, roof, open pit and rock bits; pining rods; augers; drive sockets; adapters; grinders, and grinding wheels. A price list accompanies the brochure.

(33) RADIOISOTOPES: *Oak Ridge National Laboratory* has issued a revised radioisotope catalog. Nearly 100 different radioactive preparations made and distributed by the laboratory are listed with specifications and prices. Irradiation facilities, procurement procedure, shipping information, special sources, service or custom irradiations, neutron-activation analysis, and waste disposal are detailed in the catalog.

(34) CHEMICAL BULLETINS: *Climax Molybdenum Co.* has a list, with descriptions, of currently available bulletins on molybdenum and molybdenum compounds. Categories covered are: chemical data, agriculture, analysis, ceramics, catalysts, and colors.

(35) METAL JOINING: A 16-page booklet on metal joining with low heat input is available from *Eutectic Welding Alloys Corp.* The low temperature process is claimed to minimize the residual stresses, cracking, warping, and distortion that may occur with the usual materials.

(36) EXCAVATOR DIESEL: Model 687C-18 by P&H division of *Harnischfeger Corp.* is a 6-cyl diesel engine for 1⅛, 1¾, and 2-yd heavy duty excavators. Bulletin Z-7 details specifications and performance features.

(37) ZEOLITE PROCESS: The advantages found through the use of a hot process-hot zeolite system in a boiler feedwater treatment unit are described in a reprint article from *Graver Water Conditioning Co.*

MAIL THIS CARD

for more information on items described in Manufacturers News and for bulletins and catalogs listed in the Free Literature section.

5

Mining Engineering **29 West 39th St.** **New York 18, N. Y.**

Not good after August 15, 1956—if mailed in U. S. or Canada.

Please send { More Information ☐ / Price Data ☐ / Free Literature ☐ } on items circled.

Name_____ Title_____

Company_____

Street_____

City and Zone_____ State_____

1	2	3	4	5	6	7	8	9	10
11	12	13	14	15	16	17	18	19	20
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41	42	43	44	45	46	47	48	49	50
51	52	53	54	55	56	57	58	59	60
61	62	63	64	Students should write direct to manufacturer.					

(38) INDUSTRIAL FIRE HOSE: Industrial Products Div. of *B. F. Goodrich Co.* has a 4-page illustrated data sheet on its line of industrial fire hose. A chemical treatment which protects against mildew, rotting, moisture absorption and freezing and an end protection for greater hose end strength are described.

(39) ELECTRIC OVENS: *Soiltest Inc.* has the Blue M line of electric ovens and furnaces featuring dual-banked modella elements. Construction is double-wall type with stainless steel exterior. Prices are included in folder 5520.

(40) DOCKS AND PIERS: *De Long Corp.* has a special technique of dock construction that is described in a 20-page illustrated booklet. Installations include piers, wharves, and drilling platforms.

(41) TRACTOR-SHOVEL: Payloader model HO tractor-shovel by *Frank G. Hough Co.* has 4-wheel drive and a capacity of 2¼ cu yd. Features include 40° bucket break-out at ground level, power controls, planetary final drives, torque-proportioning differentials.

(42) ROLL CRUSHERS: *McLanahan & Stone Corp.* has an 8-page bulletin on McLanahan double roll crushers for finer reduction of medium size feeds. Tables for determining crusher capacity for various roll setting and size combinations are included.

(43) NICKEL PRESSURE VESSELS: Bulletin 24 from *International Nickel Co.* summarizes important application requirements on adapting eight commercial high nickel alloys for pressure vessels.

(44) GRINDING RODS: The diamonite grinding rods by Diamonite Products Div. of U. S. Ceramic Tile Co. are specified in a new folder. Lower costs and longer service are claimed.

(45) BELT CONVEYORS: Pre-Bilt sectional belt conveyors with capacities up to 1500 tph are described in

Link-Belt Co. book 2579. Belt widths are 18, 24, 30, and 36-in., and drives range up to 40 hp.

(46) WIRE ROPE: Wire rope recommendations for industrial services are given in 24-page bulletin DH-128D from American Cable Div. of *American Chain & Cable Co.* Recommendations cover rope diameters, construction, preforming, lay, grade, and core. Charts on weight and breaking strengths are provided.

(47) TRANSISTORIZED COUNTER: Model 411 by *Universal Atomics Corp.* is a gun-shaped geiger counter employing seven transistors and powered by two standard flashlight cells. Audio amplifier, reversible shield, chromed case are some of the features.

(48) SERVICE PIPE: *Johns-Manville* has a 12-page, illustrated brochure on an asbestos cement pipe. Transite pipe is pressure formed, corrosion resistant, and features Ring-Tite coupling to provide flexibility and simplify installation.

(49) CO_2 ANALYZERS: *Hays Corp.* has 4-page catalog 56-553-1 on combustion meters used as an index to the efficiency of fuel burning operations. The meters can be provided to record draft and flue gas temperatures.

(50) AUTOMATIC CONTROLS: Catalog D-31 from *Automatic Temperature Control Co.* is a condensed guide to automation components control systems. Differential transformer systems, time controls, valves, and counters are included in the illustrated listings.

(51) CONCRETE GUNNING: A 16-page catalog by *Air Placement Equipment Co.* has the answers to some often-asked questions about air placed or gunned concrete. The spraying of concrete is desirable for unusually shaped areas as forming is often eliminated.

(52) VIBRATING SCREEN: Model S by *Allis-Chalmers* is vibrating screen in sizes up to 4x10 ft for handling of specification aggregates up to four in. and coal up to six in. Screen has bolted, lock-nut construction with close-spaced clamping bar bolts and springs.

(53) DRILLING EQUIPMENT: *Acker Drill Co.* offers a line of soil sampling tools, rotary earth augers, diamond and shot core drills, earth boring equipment, drilling accessories and supplies. Bulletins include on-the-job photos. Diagrams supplement detailed specifications.

(54) BI-METAL WIRE: Alloy Metal Wire Div. of *H. K. Porter Co.* has a bulletin on nickel-clad copper wire. It is said to provide high conductivity with resistance to oxidation, flaking, and embrittlement at high temperatures.

(55) URANIUM CONCENTRATION: A flowsheet on the "Resin In Pulp" process for uranium concentration is offered by *Denver Equipment Co.* To date, the process is not applicable when vanadium must also be recovered.

(56) HARDNESS TESTERS: Wilson Mechanical Instrument Div. of *American Chain & Cable Co. Inc.* has a 12-page bulletin on Tukon testers. These testers make possible microhardness testing using extremely shallow indentations. They may be used on fine wire, small precision parts, thin metal, jewels, plastics, glass.

(57) CASTINGS CHART: A designation chart for corrosion resistant stainless steels is available from *Empire Steel Castings Inc.* The chart correlates the various stainless types for selection and reference.

(58) AIR TOOLS: A 68-page, second edition of No. 44 catalog by *Thor Power Tool Co.* lists a line of mining and contractor's tools. Attachments, accessories, and replacement parts are included with tables of specifications.

FIRST CLASS
PERMIT No. 6433
Sec. 34.9 P.L.&R.
New York, N. Y.

BUSINESS REPLY CARD
NO POSTAGE STAMP NECESSARY IF MAILED IN THE UNITED STATES

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MINING ENGINEERING

29 WEST 39th STREET

NEW YORK 18, N. Y.

Seminole Rock Products of Miami, Fla., bought five CAT* DW15s with W15 Wagons on the basis of its previous experience with Caterpillar products. Dragline loaded with lime and coral rock, these units make a 1650-ft. haul (with short 10% adverse grade) over excellently maintained roads to the processing plant, and 2000-ft. return. After the Caterpillar DW15s had been on the job long enough to show what they could do, here is what General Manager L. G. Bunnell had to say:

"These DW15s have power to spare. With an average load of 15.2 yd. (45,600 lb. actual weight), haul and return time is only 4.09 minutes. The five units haul 358 cu. yd. per hour—estimated hauling cost is only 9.2¢ per yard. Thanks to their powerful Cat Engines, the DW15s started up from the dragline in second gear, high range.

"We are particularly satisfied with their extreme economy. In more than 1600 hours of combined operation, these five new Caterpillar DW15s have had no down time. Their rugged construction and trouble-free operation are important factors in cutting costs. Important, too, are their ease of maintenance and their ability to deliver full, foul-free power on No. 2 furnace oil.

"We like the high maneuverability of the Cat DW15 and W15 Wagon," Mr. Bunnell concludes. "This, with their dependability and power, speeds up our operation considerably."

Your Caterpillar Dealer—who provides skilled service and factory parts you can trust—will demonstrate the DW15 on *your* job. Give him a call soon.

Caterpillar Tractor Co., Peoria, Illinois, U.S.A.

CATERPILLAR*

*Caterpillar and Cat are Registered Trademarks of Caterpillar Tractor Co

**NAME THE DATE...
YOUR DEALER
WILL DEMONSTRATE**

Virtually all mining areas now standardize on Clarkson stainless steel units to accurately feed reagents to flotation cells. In 1939, Miami Copper Company, Miami, Arizona, was first of the Copper Concentrators to install stainless steel feeders. These early units still provide maintenance-free service for Miami Copper.

Stainless steel units good as new after 17 years of corrosive reagents

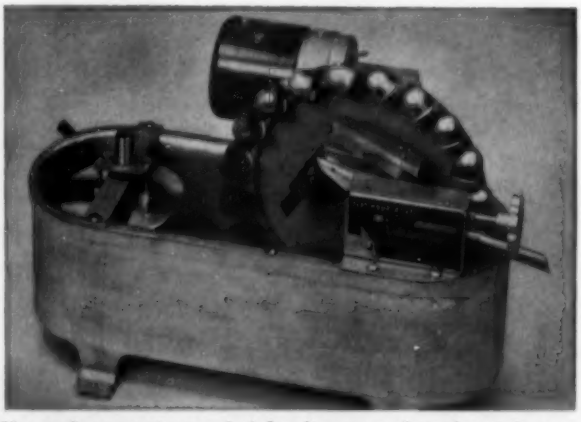
FLOTATION is an operation where the earning-power of stainless steel comes to the top.

Reagent feeders operating since 1939 still give good-as-new performance because they're made entirely of austenitic chromium-nickel stainless steel.

Regardless of where you operate, whether in nonmetallic or metallic mining, there are hundreds of applications where chromium-nickel stainless steel will pay off handsomely.

As in feeders, stainless steel containing nickel lengthens equipment life. It resists attack by chemicals and atmospheres over a wide range of concentrations, pressures and temperatures. What's more, it is strong, tough, and ductile. Its high resistance to impact, wear and abrasion spells long, maintenance-free service under severest operating conditions.

There's no need to put off using stainless steels where you want long, money-saving performance. Leading steel companies produce austenitic chromium-nickel stainless steels in all commercial forms.



No replacements needed despite years of service under corrosive and abrasive conditions on units such as this. View shows interior of durable, corrosion-resistant 18-8 chromium-nickel stainless steel feeder, developed and produced by Clarkson Co., a Division of Equipment Engineers, Inc., San Francisco.

INCO

THE INTERNATIONAL NICKEL COMPANY, INC. 67 Wall Street New York 5, N.Y.

Borax Mine to Become Open Pit

The Pacific Coast Borax Co. has commenced an $18 million development program at Boron, San Bernardino County, Calif. In January work got underway on stripping of overburden for the conversion of the underground operation into an open pit. The project, expected to be completed next year, includes the construction of new refining and concentration plants.

Reynolds to Begin Bauxite Shipments from Haiti

In May, Reynolds Mining Corp. expects to complete construction work at its mines and ore handling facilities in Haiti and commence shipping bauxite to its La Quinta alumina plant at Corpus Christi, Texas. Reynolds is now operating two bauxite ships between Jamaica and the Gulf ports and has two additional carriers under construction.

New Fuller's Earth Plant in Operation

Near Meigs in southwest Georgia, Waverly Petroleum Products Co. of Philadelphia, Pa., has put into operation a new plant to process fuller's earth for use as oil and grease absorbants. Mining is carried out by stripping methods in this area, the heart of a rich fuller's earth district.

Uranium Merger Proposed

During April stockholders of Sabre Uranium Corp. are meeting in Dallas, Texas, to consider the question of the proposed merger with Piñon Uranium Corp. The approval, which appears likely, will bring together two large groups of rich uranium claims in the Ambrosia Lake district near Grants, N. M. Stockholders are also considering a proposal whereby American Metals Co. would receive 25 pct interest in the new firm in return for an investment of $4.5 million, largely for the construction of a 1500 tpd mill near Grants. American Metals Co. would also direct the development of the merged firm's uranium properties.

Mineral Laboratories Organized in Missouri

During April the Geomar Mineral Research Laboratories, a division of the newly formed Geomar Minerals Corp., was organized at Rolla, Mo. The unit will consist of five integrated laboratories for petrographic and mineral analysis, X-ray, spectrographic, chemical, and radiometric analysis. It will be the only operation of its kind between Denver and Pittsburgh.

In Nevada...And the World Over

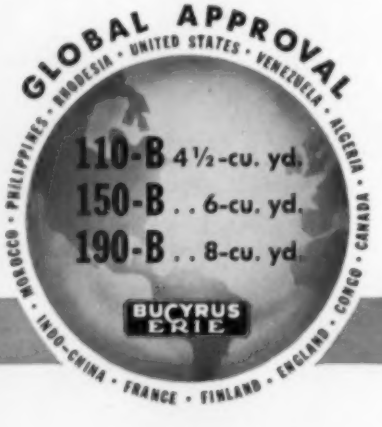
Bucyrus-Eries get down to paydirt FAST

IN open-pit mines everywhere, Bucyrus-Erie shovels like this 6-yd. 150-B are demonstrating their ability to strip overburden and load ore fast and economically.

You can see *how* they do this as you watch them work — heaped dippers, smooth cycles, ease of control that cuts operator fatigue, plus year-after-year dependability.

You can see *why* Bucyrus-Eries are superior performers when you check their design and construction. The strong yet light front end reduces deadweight, saves power. Ward Leonard electric control provides immediate response, permits time-saving coordination.

We would be glad to tell you more about Bucyrus-Erie excavators — how they can improve *your* operations.



BUCYRUS - ERIE COMPANY

SOUTH MILWAUKEE, WISCONSIN

97L56C

Canyon wall exposes the Wingate formation.

Airlift Compressor To Drilling Site

The Utah Rockies in twenty below zero weather at an elevation of 7000 ft—the problem—set an air compressor in place at the top of a canyon wall with sheer 1500-ft sides.

Atlas Corp. faced this difficulty when they decided to test drill for uranium in the Hidden Splendor area of Utah. They hit on the solution of dismantling a LeRoi 105 compressor and flying it, piece by piece, to the selected spot on the Wingate formation, by way of helicopter.

When the compressor was reassembled, its operation proved the airlift worthwhile and several drill rigs were used at sites some distance from the compressor area.

Helicopter used in transporting hovers over reassembled compressor.

Survey With New Airborne Detector

A new electromagnetic detector, trailed from a plane by means of a cable, is being used as an exploration tool by Aerophysics of Canada Ltd. The device is intended for quick location of areas of heavy sulfide deposits, principally copper and lead, so that extensive ground surveys may be concentrated on especially promising sections.

Recordings are made from the output of a receiver which is slung from the plane in a *bomb* at the end of a 500-ft cable. The detector can cover areas at speeds of over 100 mph. and cost is estimated at one tenth that of ground surveys.

The electromagnetic system uses a wire coil to create an alternating field which induces a secondary field from a conductive body coming within range. High sensitivity to sulfide ore and good discrimination against overburden is claimed in field tests.

As the bomb trails about halfway between the aircraft and the ground, complete records are taken of the output of the electromagnetic equipment, the aircraft altitude, and simultaneously aerial photographs are taken of the ground below.

It is indicated that the usefulness of the system depends on regional makeup, and exploration will be best done after study by geologists and geophysicists. Graphitic zones and subsurface saline areas can also cause anomalies.

Aerophysics will launch its third detector-equipped plane late this year.

An Anson aircraft carries the transmitter and recording equipment. Anomaly detector is towed in bomb at the end of a 500-ft cable.

Technician keeps a constant check on his instruments as plane cruises at low altitude.

Largest Titanium Plant Now in Production at Ashtabula, Ohio

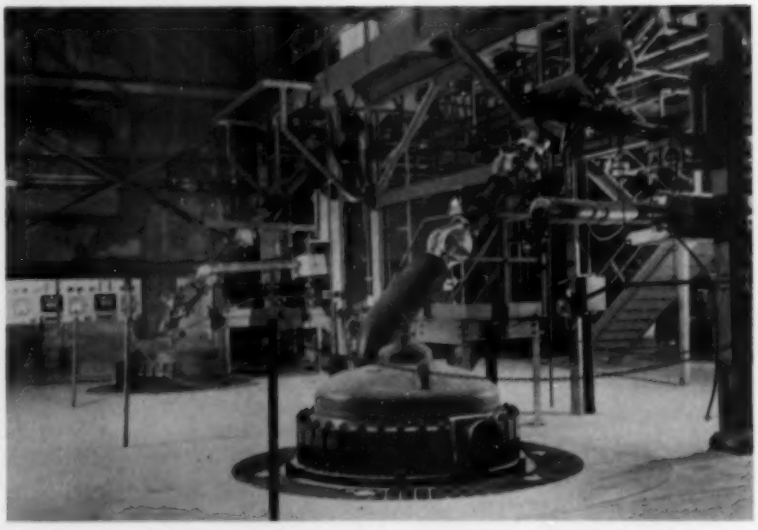
Sodium Reduction Method to Yield 7500 Tons Annually

A six-year program costing over $2 million resulted in the production recently of the first heat of titanium sponge at the new Ashtabula, Ohio plant of Electro Metallurgical Co., a division of Union Carbide & Carbon Corp.

The privately financed plant is the largest titanium producing facility built to date and has a rated annual capacity of 7500 tons of titanium sponge.

Using a sodium reduction process, the new plant is the first to depart from the magnesium reduction method that has been used in all previous commercial production. The company has had this process in operation on a pilot and prototype plant scale at its Metals Research & Development Laboratories in Niagara Falls, N. Y.

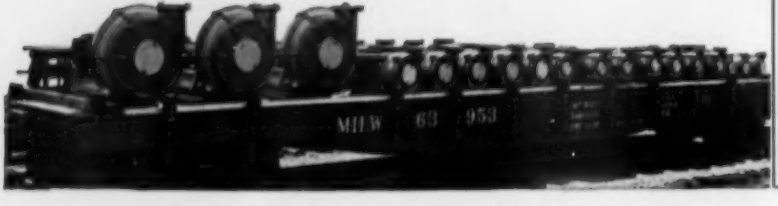
Titanium is about half as heavy as steel though its alloys have tensile strengths approaching those of most steel alloys. Strengths are maintained at temperatures up to 700°F and corrosion resistance of the metal is high. These qualities make the metal especially suited for use in high-speed planes and jet engines. Electromet also sees large potential civilian markets in chemicals, petroleum, food, electronics, and atomic energy.

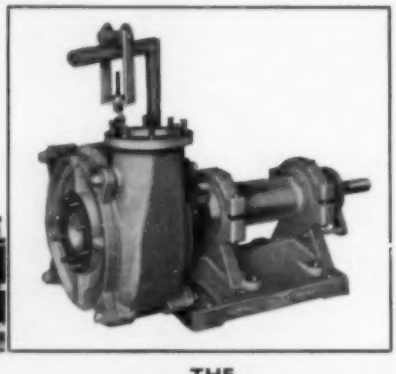


Equipment used in the production of titanium metal sponge at the Electromet plant at Ashtabula, Ohio. Plant is expected to reach rated capacity of 7500 tons per year by the fourth quarter of 1956.

From 3 tons in 1948, titanium production mounted to 7200 tons last year. Total production in 1956 is expected to reach 13,000 tons.

A truckload of economy for a base metals concentrator in Northern Quebec...





Black Diamonds from Newcastle . . . precious stones of Kimberley . . . in fact anywhere in the world where engineers *have solved* the problem of pumping abrasive solids in suspension *economically*—you will find the Linatex Pump at work.

Here is the inside story. When we talk of the Linatex Pump, we must talk of Linatex around which, quite literally, it is designed. This living, natural rubber, is stabilised by a patent process, giving it greater resistance to abrasion than any known material. Being resilient, it 'bounces off' particles without damage to itself. Thus by lining our pump with Linatex we greatly reduce wear. Also the impeller, of solid Linatex, is virtually indestructible. Compact and easily accessible for routine inspections the pump also features a patented hydrostatic gland and alternative types of packed glands.

THE LINATEX **PUMP** *the most lasting pump in the world*

6 of the 15 Linatex factories in the world
Any of them will see that your enquiries receive energetic action

AUSTRALIA		U.S.A.
Linatex (Australia) Pty. Ltd., David Street, Dandenong, Victoria, Australia		Linatex Corporation of America, Vernon Avenue, Rockville, Conn. U.S.A.
CANADA	**ENGLAND**	**MALAYA**	**STH. AFRICA**
Wilkinson Linatex Co. Ltd., 2210 Parthenais St., Montreal 24, Quebec, Canada	Wilkinson Rubber Linatex Ltd., Camberley, Surrey England	The Wilkinson Process Rubber Co. Ltd., Batu Caves, Selangor, Fed. of Malaya	R. J. Spargo Ltd., P.O. Box 7128, Johannesburg S. Africa

P 8 (M)

CABLE AND MINE EXPERIENCE enable Anaconda to design and build shovel cable that resists razor-sharp rocks, water, sun, heat, cold, and — most important — run-overs.

This cable was "built" in our mines to perform better in yours

If you were a cable manufacturer as well as a miner, you could use your mine experience to build cable better than others.

But since you aren't, it will pay you to buy your cable from Anaconda—who *is* both cable manufacturer *and miner*.

FOR SHOVEL CABLE

Years of mine experience . . . of cable testing and improving . . . have taught us how to make a cable that *stays on the job longer under all conditions*. It's Securityflex* Type SH-D cable.

New, job-tested, high-grade butyl insulation assures better resistance to heat, water and compression. And new neoprene jacket makes this cable even tougher and more abrasion-resistant.

For Type SH-D, *or other shovel cable*, let Anaconda mine and cable experience pay off for you. See your Anaconda Distributor. Anaconda Wire & Cable Company, 25 Broadway, New York 4, N. Y. 55021

*Trade Mark

ANACONDA®
MINE CABLE

FLAT-TWIN CABLE

Improved stranding, new insulation, new grounding wire, and neoprene jacket make this a superior cable for shuttle cars, continuous miners, loaders, drill trucks, cutters.

POWER CABLES

AnacondA Types W & G are rugged, sturdy and long-lived. Used for mine power, shovels, continuous miners, loaders, drill trucks, cutters.

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Unexcelled for strength, wear resistance and long life. Type SO (heavy-duty) provides superior service on remote control and hand drills.

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169

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Ground Fall Accidents Reduced

The National Safety Council reports that the 1955 International Campaign for the Prevention of Falls of Ground Accidents proved the effectiveness of pointing up the value of common safety measures. An estimated 260 injuries were prevented last year, of which a probable 15 would have been permanent disabilities and fatalities.

A 29 pct reduction in the frequency of injuries from falls of ground in comparison with the average for the previous two or three years was the positive result of the campaign. Three-fourths of the participants either reduced the incidence of ground fall accidents or had no accidents attributable to this hazard. Awards for attaining the campaign goal of 50 pct accident reduction were won by 122 mines.

No unusual cure was employed but the campaign forced attention on the need for safety, created interest in finding a solution, and encouraged action. While the danger of falls of ground may never be entirely eliminated, the council points out, safety precautions have proved their effectiveness in real results.

President Acts on Engineer Shortage

A National Committee for the Development of Scientists and Engineers was established by President Eisenhower in early April to "foster the development of more highly qualified technological manpower." The committee is composed of 19 leaders in engineering, science, education, management, labor, state and local government, and the humanities.

Formation of the group was the result of a study headed by A. S. Flemming, Director of Defense Mobilization. The study group maintained that main responsibility for increasing the number and quality of technicians lies with citizen groups organized to act effectively. The new committee is expected, by presidential direction, to enlist cooperation "in analyzing the problem and developing programs to deal with it, and to take the lead in coordination of interested organizations outside the Federal Government."

Among those appointed to the national committee by the President was Thomas H. Chilton, president of the Engineering Joint Council, who will act as a representative of the engineering interests.

Recent reports from a subcommittee of the Congressional Joint Atomic Energy Committee indicate that the Soviet Union already has more engineers and scientists than the U. S. and that Russia is graduating more than twice as many each year as this country.

● Mining contractors, ore prospectors, coal operators and construction firms are realizing tremendous savings by taking advantage of our **exclusive fabrication service!** Contractors send us the necessary diamond stones from their own stocks—we hand set them in a super-hard tungsten carbide crown and braze to the threaded steel blank. Hand-set bits assure the proper positioning of each diamond stone to achieve maximum cutting efficiency. The carbide matrix holds the diamond stones until entirely used up. These advantages mean lower drilling costs to you. We can also supply complete core bits or salvage the stones from used bits at nominal cost. Supplied in standard sizes EX, EXE, AX, BX, NX, etc.

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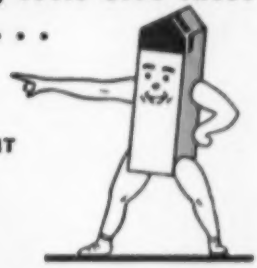

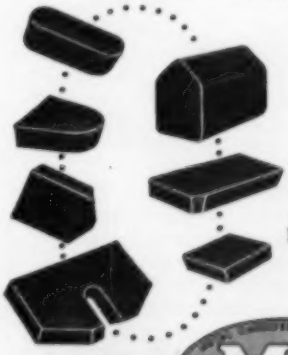

● A complete line of low-cost, high-quality Talide Tips is offered fabricators and users for tipping machine bits, rock bits, drill bits, roof bits and open-pit bits. All Talide Tips have a special surface finish that facilitates brazing. Non-standard shapes and sizes quoted on request.

DIAMONDS are now being produced so successfully in a pilot plant operation that "man-made diamonds will have an important impact on American industry and defense," an official of General Electric Co. predicted at the presentation to the Smithsonian Institution of Washington, D. C. of a cluster of the first diamonds made in the GE research laboratory in Schenectady, N. Y. in 1954. These were surrounded with a 100-carat sample of diamonds subsequently made in the pilot plant operation at GE's Carboloy Dept. in Detroit.

The diamonds are made by subjecting "a carbonaceous compound" to pressures up to 2.7 million psi and temperatures to 5000°F for extended periods. Steady progress has been made in bridging the gap between the laboratory and the production line with emphasis placed on the achievement of a marketable price. No attempt is being made to produce stones of gem quality and size, but manufactured diamonds have proven to be identical with those mined from the earth.

Diamonds are mined principally in Africa, with a smaller supply from South America, at the rate of some 2½ tons annually. The major portion is for industrial uses with the U. S. as the largest importer. With industry of this country completely dependent upon foreign sources, the development of diamond manufacture takes on particular significance. Fortunately, some 80 pct of the industrial diamonds now used are of sizes that can be produced by GE's process.

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A STUDENT engineer is a big fellow with a steel-trap mind who learns his job and not much more. True or not that is the impression formed in the minds of individuals who see a danger in the ever increasing specialization of engineers. Is their face well-founded? Certainly the engineer must rigorously apply himself to a particular study. The day of the jack-of-all-trades will never come to nuclear engineering or electronics. It would be a man with a poor science indeed who could claim himself its master in every phase.

But the engineer is often in a position of great social responsibility and his liberal education is of vital importance. Can an engineer, whose career training demands so much, also acquire the liberal education which means the inclusion of the humanities and social sciences in his college curriculum? The American Society for Engineering Education after three years of study insists that the engineer not only can but must do so.

A special committee composed of distinguished teachers in engineering schools and representatives of industry found that "the humanities and social sciences are, in a deeply serious sense, practical and useful." They are not speaking of "a cultural veneer designed to make the engineer acceptable in polite society." Instead they maintain that liberal arts courses "contribute to professional competence in the broad sense of enabling the engineer to see his own activities in their human and social contexts, to understand his own objectives, methods, and problems in relation to those of men engaged in other activities."

Leading schools have shown that carefully planned programs will reinforce engineering training while providing sound instruction in the arts and social sciences. There is an area in the liberal program which gives so far unexplained difficulty to engineering students. While the reception given to non-literary arts is good, the emphasis found needed on English and composition courses still shows a "staggering discrepancy" between effort and results. There is a steady stream of complaints that graduates cannot express themselves in speech or on paper. The study suggests that "Engineers might well take the initiative in sponsoring a thorough-going and hard-headed study of the whole matter."

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MAN or Mob? The genius or the group? That is the sort of choice we are facing if we heed the words of Crawford Greenewalt, president of du Pont. Speaking at a meeting of the American Newspaper Publishers Assn. in New York, Mr. Greenewalt chose as his theme the uncommon man, and termed him the key to progress.

Core of the problem in preserving and organizing progress, (ordinarily a rather disorganized affair), is how to preserve the creative genius of the uncommon man despite the emphasis upon group effort in modern industry. "Behind every advance of the human race," he said, "is a gem of creation growing in the mind of some lone individual, an individual whose dreams awaken him in the night while others lie contentedly asleep. We need those dreams, for today's dreams represent tomorrow's realities."

Without putting words in the mouth of the man from du Pont, it would suffice to say that he also seems to see virtue in implementation of dreams by group hands. The problem: how to have the group organization without killing the special kind of wild geese that lay the golden eggs for the future.

After paying his respects to the value of the uncommon, and creative, individualist, Mr. Greenewalt took a few gentle swipes at the tired adjectives still being used to describe businessmen in general. He feels that the stereotype of the big, bad, and rugged businessman is at best dated, at worst completely untrue. Looking back 50 years he notes the change from the colorful, personable, and somewhat rugged individuals then dominating the scene to the relatively quiet anonymity of the group now characteristic of the business world. Personally he takes the mislabeling in stride.

"I might record the fact that I started many years ago as a scientist—hence, at that time, was presumably long-haired, and doubly so since I was also an amateur musician. Imagine how disconert-

ing it was, some years after having made the transition from the laboratory to the business office, to find myself characterized by no less an agency than the Dept. of Justice as a "monopolist by marriage,' and by the *Daily Worker* as a bloodless capitalist. Time Inc. was kinder—calling me merely 'graying, hawk-nosed,' a term my children have adopted with some glee." Mr. Greenwalt may be graying, but he hasn't lost his sense of humor.

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ONE way to solve the shortage of qualified teachers and bring top-flight technical men from the field to the classroom has been found by the University of Utah. By getting cooperation from industry the University has brought in a series of outstanding lecturers to the school—men from companies such as Kennecott, U. S. Smelting, Combined Metals, Anaconda, Asarco, Hercules Powder, the U. S. Public Health Dept., and the Independent Coal & Coke Co.

To deal with the topic of smelter schedules the school drew upon the practical background of U. S. Smelting, Refining, & Mining Co., as well as upon the personal experience of Blaine Watts, company ore purchaser. Stan Michaelson, chief engineer for Kennecott's Western Mining Div. spoke on "Engineering Mining Valuation" and "The Design of Mine Plant". Others handled topics equally up their alley.

Bringing these men to lecture at the University has brought enthusiastic reception from the students, faculty, and the mining fraternity. No increase in faculty. No cost to the school, and the students get the facts.

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TEN million is a lot of almost anything these days—except in a government budget and 10 million was the number of feet of hole punched in by Canadian diamond drillers last year according to a report in the *Northern Miner*. Without speculating on the precise cost of this much hole, it is interesting to correlate some figures from an address to the shareholders of International Nickel Co. by John F. Thompson, Inco board chairman.

In ten years Inco has shelled out $31 million for exploration, distributed as follows: $2.5 million for geophysical work, largely airborne; $11.5 million for surface diamond drilling; and $17 million for all other exploration. It is pretty risky to project such a slim slice over the whole picture but if other firms approach anything like the ratio shown by these figures, that 10 million ft of diamond drill hole for 1956 is the index of an astronomical exploration expenditure just north of the border.

Geophysics is no longer the infant of the exploration field,—not at $2.5 million. That sort of figure is getting into the area of expenditure more characteristic of petroleum exploration.

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DARRELL Huff wrote a book *How to Lie with Statistics* (W. W. Norton Inc., N. Y.) as if a book on that subject were just what we needed today. Actually Mr. Huff lives up to his name by poking some not so good natured fun at the man with a number and a purpose—and no scruples. To give a sample of the slant: "The number of (television) sets in American homes increased around 10,000 pct from 1947 to 1952. Project this over tne next five years and you find that there'll soon be a couple of billion of the things, Heaven forbid, or forty sets per family." Take your tongue out of your cheek now, please.

Men have made a career for themselves by just hollering "give me the figures" often enough and loud enough. (The rule here is never quite accept any set as what you wanted in the first place). On the serious side the problem of loose numbers, or if that stretches the language too much for the purists, loosely quoted numbers, is a real problem these days. Enough so to make the Royal Bank of Canada devote a whole monthly letter to the subject. Complete with selected reading list it organizes cautions on a variety of points.

As the Royal Bank points out it is not enough that just the statistics themselves be honest (or we would say the statistician), intelligence in use must be the added ingredient. As they phrase it, "They (statistics) are a base upon which to formulate sound business judgment: that is all."

In concluding, the letter quotes from Mr. Moroney's *Facts from Figures*: "If you are young, then I say: Learn something about statistics as soon as you can. Don't dismiss it through ignorance or because it calls for thought . . .If you are older and already crowned with the laurels of success, see to it that those under your wing who look to you for advice are encouraged to look into this subject. In this way you will show that your arteries are not yet hardened, and you will be able to reap the benefits without doing over much work yourself. Whoever you are, if your work calls for the interpretation of data, you may be able to do without statistics, but you won't do so well." Amen.

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AEC wants you. With an increasing number of private companies negotiating to construct and operate uranium mills, the Commission is in need of additional personnel in its Processing Div. to administer the program and necessary contracts. Interested graduate engineers with a minimum of five years experience in the design, construction, or operation of ore beneficiation plants should contact the Personnel Office, U. S. Atomic Energy Commission, Grand Junction Operations Office, Grand Junction, Colo. Annual salary range is $6,390 to $7,570 depending on qualifications and experience.

POWERHOUSE ...one D397 does it all!

In one of the world's largest phosphate beds, near Soda Springs, Idaho, this CAT® D397 Electric Set is the main source of power for the crushing plant shown here. Output of the crusher, owned by Morrison-Knudsen Co., Inc., is 1000 tons per eight-hour day of 1½"—⅜" quartzite. The crushed quartzite is raw material used in Monsanto Chemical Company's nearby elemental phosphorus plant.

The D397 Electric Set furnishes power for a 125 HP motor on the Simmons cone crusher, a 100 HP motor on the Cedarapids jaw crusher, and nine motors totaling 172½ HP on conveyors and vibrator screens.

Now there is a new Caterpillar D397 Electric Set with Turbocharger, developing 350 KW. It is self-regulated for easy, adjustment-free operation and steady voltage. It incorporates Caterpillar's amazing new generator, which exactly matches the D397 Engine powering it.

Reliability and long life are important in an electric set, which is often called upon for continuous operation. The Caterpillar D397 has both, with such features as aluminum alloy bearings, "Hi-Electro" hardened crankshaft and cylinder liners. It also has highly effective oil and air filters designed to protect the engine's Caterpillar-built precision under even the dustiest conditions.

There is a complete line of Caterpillar portable and stationary Electric Sets, up to 350 KW. Your Caterpillar Dealer — who provides skilled service and parts you can trust — has full details on the engine or electric set best suited to *your* operation. See him today.

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MODERN HEAVY-DUTY ELECTRIC SETS

Regional Meetings Broaden Viewpoint

THIS month members living in the Northwest and those who could travel out there had an opportunity to take a look at the whole of the mineral industry picture. May 3 to 5 marked AIME's Fifth Pacific Northwest Regional Metals and Minerals Conference. What is significant about this meeting, aside from the high caliber of the papers, is that this is a meeting place on a regional basis for technical interests in two branches—Mining and Metals.

Again, in September, those members in the Rocky Mountain province and those who can go out there will have another of these opportunities to see the large picture. This occasion will be the Rocky Mountain Minerals Conference on September 26 to 28 at Salt Lake City. Those guiding the gathering stress that interests covered will cut across the fields of Mining, Metals, and Petroleum. Last year the sessions at this conference were set up on topics of joint interest, drawing men from all technical divisions into discussion of common problems.

This process of cross fertilization has its surprises, some of which show up during discussion. Men in diverse fields are often hacking away at a problem without knowing that they are on opposite sides of it. Meetings organized on a topic or regional basis are the place to find this out.

It is far from new to use common problems to draw men of varied specialization into joint discussion. Certainly the title of one of the Iron and Steel Div.'s conference committees is a case in point: Blast Furnace, Coke Oven, and Raw Materials. In addition to this range of interests, there is the Open Hearth Committee meeting simultaneously with the BFCO&RM group at an annual spring conference. This particular meeting gives participants a chance to take a look at the whole field of iron making—from coal, limestone, and iron ore down to the final ingot.

The preliminary announcement for the AIME Northeastern Mining Branch Conference at Hershey, Pa., November 8 to 10 shows the same kind of thinking. The early information is that the technical program at Hershey will be "comprehensive, branchwide."

Still another approach to this problem is evidenced by the annual Electric Furnace Conference, set this year for December 5 to 7 in Chicago. Here the process is the topic, and again the specialized problems fit into a broad economic and industrial framework.

Looking Ahead for the Institute

In June, MINING ENGINEERING will carry the report of the AIME Long Range Planning Committee, together with some explanatory material and comment. Copies of this report are available now for those who write to headquarters in New York. We wish to stress the importance of the questions this type of planning raises, and hope that every reader will take a close look at the report when it appears in June. Looking far ahead is something that individuals and organization do all too seldom, and this underlines the importance of those times when it is done.

In a way this report on long range planning might be termed: "the engineer looks at his professional engineering society." The same issue will also carry something of a different nature that might be termed: "the engineer looks at his profession." Of this, more below.

Professionalism and the Engineer

Some time ago Engineers Joint Council (of which AIME is a member), charged a committee with the task of studying professional employment conditions and preparing information for employers on the effect of various factors on the development of a truly professional outlook by engineers. This work has resulted in a report now being released by the EJC Board, entitled "Professional Standards and Employment Conditions."

One benefit, at least, of the shortage of engineers—or if one wishes, the lack of an oversupply of engineers—is that the profession and members of the profession are getting more attention both within and without the field than ever before. For once the vital role played by the profession is being recognized by management, by educators, and by the public.

Benefit to the profession is almost certain to flow from this increased notice and attention. Among the hoped-for gains: better training for engineers, more recognition, more financial recognition, and particularly, a stronger professional spirit. This EJC report should help in several of these areas. At least some observers feel that now is the time to capitalize on a favorable situation and raise the stature of engineering as a profession, and to encourage the growth of professionalism, particularly among the younger practitioners.

Employers will play a role in this growth of engineering, it is felt, particularly since many are now searching for other solutions to their professional manpower problems than merely raiding their competitors. Management in many cases appears eager to find assistance and answers as to problems of proper treatment of professional level employees. Several firms have received rude shocks when staff attitude toward the company was revealed through questionnaires.

When one large firm moved a laboratory it was jarred to find how few of its people moved with the lab. Money was not the problem. Checkbacks on the people who left revealed big areas of dissatisfaction. Lack of recognition was a sore point, as was lack of a clear picture from the company of where they fitted into its overall operations.

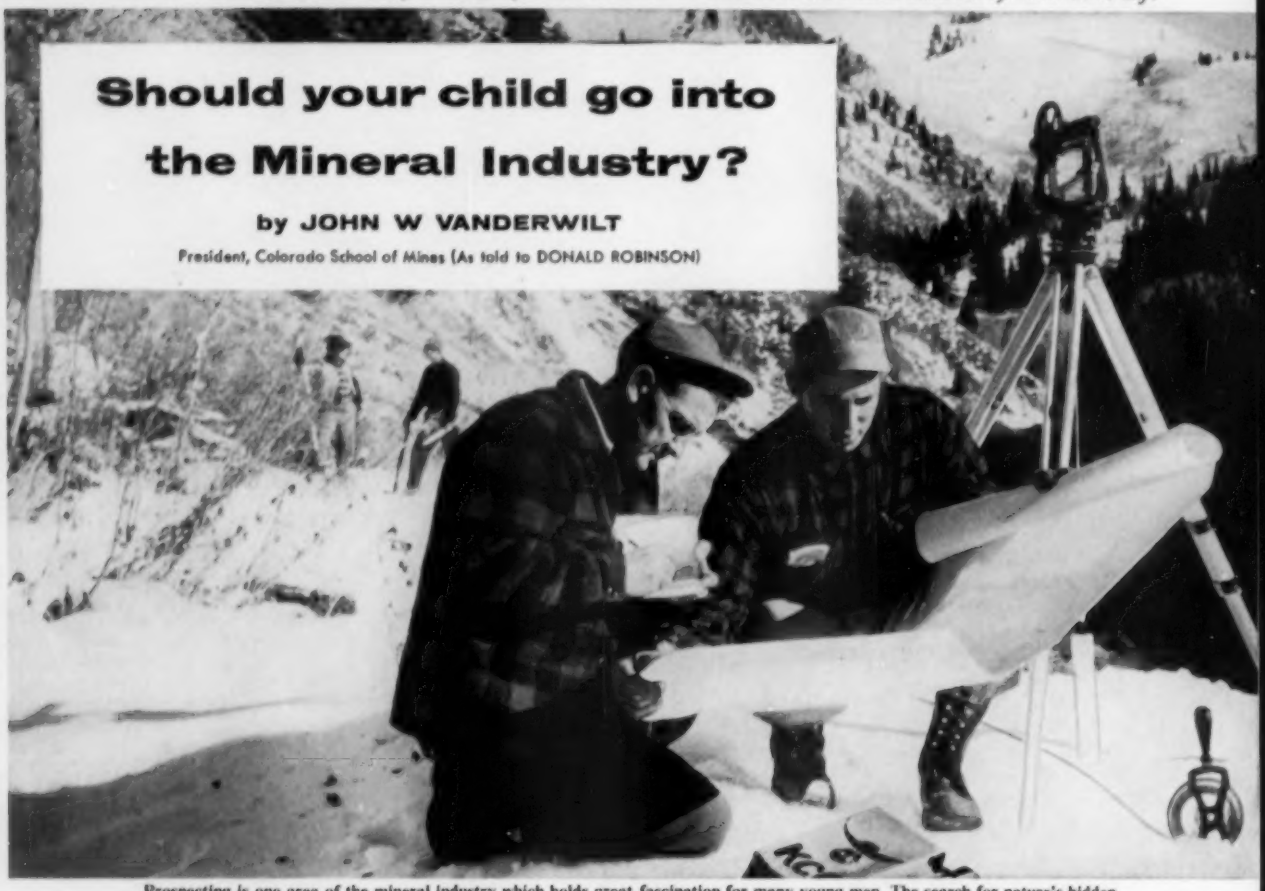
Everyone—government, business, or college—is looking for engineers right now. Out of this search are not only going to come constructive action for the future, but much closer attention to what is now available—to the conserving of the available professional engineering manpower resources they do have.

R. A. Beals

Should your child go into the Mineral Industry?

by JOHN W VANDERWILT

President, Colorado School of Mines (As told to DONALD ROBINSON)



Prospecting is one area of the mineral industry which holds great fascination for many young men. The search for nature's hidden resources often brings a geologist or geophysical engineer unusual opportunity for travel, adventure and rugged outdoor living.

I'VE HEARD it said that the day of the pioneer is gone. That we Americans have turned soft. That, somewhere down the line, we've lost our old-time taste for exploration and adventure.

"There just aren't any more Davy Crocketts or Dan'l Boones left," people tell me. It isn't so, though.

At this very moment, Americans are making their way into some of the wildest, most remote corners of the world, with the same disregard for danger and discomfort that our pioneer-ancestors showed.

I know of Americans who are daring the backwaters of the Amazon; others who are scaling the Andes, cutting through the jungles of Africa, crossing the barren deserts of Asia, battling the snake-infested swamps of our own southland. And, without fear of contradiction, I can say they're relishing every minute of it.

"Never had it so bad, and never enjoyed anything so much," one of them wrote home.

These young men—as well as their colleagues in more settled mines, mills and laboratories—are trail blazing along a never-ending frontier: the mineral industry. Regardless of difficulties, they are finding precious minerals, extracting them and then converting them into their most useful, valuable forms.

It is a vital job they are all doing.

Our national economy is rooted, in large part, in mineral production. Take coal. Without it, much of America's industry would grind to an immediate halt. Without oil, no plane would fly, no automobile would run. Without copper, every electric generating station would shut down. Without uranium, our national defenses would be crippled. And the same thing is true for dozens of other minerals.

Recently a friend of mine remarked to me, "Our entire civilization would crumble if it weren't for minerals and the scientists and engineers who make them available to us." He was not exaggerating the situation one whit.

There are several different categories of scientists and engineers in the industry. In brief, this is what they do.

The geologist deals with the structure of the earth. In particular, he specializes in the properties, occurrence and uses of minerals and rocks.

The geological engineer covers much the same area, but he pays more attention to the practical applications of the science and less to basic research.

Next is the geophysical engineer. He seeks hidden indications of mineral wealth by means of seismic, magnetic and other new, almost magical methods.

The mining engineer is responsible for taking the raw materials from the earth.

Now comes the metallurgical engineer and he has a vast scope, too. He not only works on the extraction of metals from their ores, but on the adaptation of these metals to man's purposes. He is constantly evolving new and better alloys or new uses for old alloys.

The petroleum engineer concentrates on obtaining oil and gas out of the earth.

The petroleum refining engineer takes up where the petroleum engineer leaves off. His job is to develop and improve the many thousands of products, such as gasoline, lubricants and asphalt, that can be made out of petroleum; or that utilize petroleum, such as nylon.

Last is the ceramic engineer. He focuses on nonmetallic minerals and the products that can be fabricated from them, such as brick, glass, enamels, electric insulators and high temperature products.

The industry in which all these professions are engaged is enormous. In the U. S. alone, the value of mineral production was, at last count, over $13,000,000,000 a year. Today, more than one tenth of all gainfully employed Americans make their living out of minerals.

For scientists and engineers, the horizons of this industry are especially wide. They can be based here or abroad. They can go out in the field and prospect for new mineral deposits or they can work in already proven mines. They can accept posts in manufacturing plants, research laboratories and government offices. Or, if they prefer, in college classrooms.

Openings for trained men are many. I say this from personal observation.

"Did you have a tough time finding a job?" I asked one of our graduates recently.

"Gosh, no," he replied. "The toughest problem I had was choosing which job I liked best."

The fact is that there is a decided scarcity of young scientists and engineers in minerals. Throughout the entire United States, the total number of graduates in this field, last year, was under 1,500. Most students at top-flight schools are offered between three and four jobs each, upon graduation.

Naturally, you want to know what the financial outlook is for mineral men. Here is the picture.

Graduate engineers can expect a starting salary in industry of $365 to $425 a month. Inside of five years, those with average ability should be drawing $7,500 a year. Ten years after graduation their earnings should be $10,000. Only the sky is the limit for very good men. Many engineers go into top management and earn exceptionally high pay. I know several who are making $50,000 to $100,000 a year.

Teachers usually begin with about $3,600 a year and top men go up to $10,000 a year. Moreover, a school year is usually nine months and some faculty men regularly double their year's income by acting as industrial consultants during their vacations.

Always, for anybody in the mineral industry who tries, there is the possibility of striking it truly rich. A few years ago, a penniless young geologist I know went prospecting for uranium. He spotted a deposit. A big one. Today it is reported that he is worth several million dollars.

This past year two of our own students at the Colorado School of Mines went prospecting for uranium in their spare time after class. They made a sizable strike within twenty-five miles of the campus.

There are also rewards in minerals beyond money. It is an exciting, adventure-filled life. The work presents a ceaseless challenge. Best of all, you feel that you are helping to create new wealth, new industries, new jobs for your fellow men.

I know that sensation at firsthand. Back in 1928, I was associated with a group of U. S. Geological Survey men sent to prospect for potash deposits. The United States was then dependent upon foreign producers for most of its potash for fertilizer and other industrial uses.

We began drilling east of Carlsbad, New Mexico. Down we went 500 feet. All we found was rock of no value. Down we went to 1,000 feet. Still only worthless rock. Deeper we drilled to 1,500, 1,800, 2,000 feet below the surface of the earth.

Up came more samples of ore. This time the samples were spotted with sylvite and bitter-tasting potash chloride. We'd hit a fine deposit.

In large part, the American potash industry—producing about two million tons a year—has grown out of that one strike. It is a source of genuine satisfaction to me.

Rewarding as the mineral industry is, you musn't think that it is all "peaches and cream." The hours are long and the conditions can be rugged. Young mineral men may find themselves working in a dilapidated lean-to deep in the Canadian bush, or on an exposed drilling platform away out in the Gulf of Mexico, or in a cramped Quonset hut off on the burning sands of the Arabian Peninsula. Even in this country, they may have to make their home in a pup tent on some jagged wasteland 100 miles from civilization. For months on end they will have no medical facilities, no recreational outlets, no comforts of any sort. Long separations from wives and families can be the rule, not the exception.

Certainly, I would not recommend the mineral industry for a girl unless she were prepared to overcome herculean obstacles. Some women have done well in geology, and a few at the research side of mineralogy, but they are still the exceptions.

Is there any way, people sometimes ask me, to tell whether a boy is well-suited for a career as a scientist or engineer in the mineral industry?

I think there is. Answer these questions honestly and you should have a fair idea of a boy's prospects.

1. Is the boy curious about scientific matters? Does he wonder about the make-up of the earth, for instance, or the broad principles underlying atomic energy? He should. 2. Is he good at mathematics, chemistry and physics? He'll need to know them through and through. 3. Is he adept with his hands? 4. Has he an analytical, broad-visioned mind? This is a field that requires clear and creative thinking. 5. Can he make up his mind quickly? He'll have to. Men's lives may depend on it.

Just recently, a young mining engineer was walking through a passageway down in a coal mine when his nose picked up a trace of gas. He didn't stop to talk. Instantly, he ordered the mine cleared. If he hadn't, eighty-odd men would have probably been fatally gassed.

6. Is the boy in good health and does he have a robust constitution? Both are vital. 7. Does he have a sense of adventure and does he enjoy outdoor life? If he doesn't, he's going to be a fish out of water. 8. Does he get along well with people? In this industry, he has to be a team player.

Now we reach the question of how a boy obtains his training for the mineral industry. It's not easy. I must tell you that.

High grades—a boy should be at least in the upper half, preferably the upper third of his high school class—are essential for entrance to a good college. And, an extensive grounding in mathematics, chemistry and physics is imperative.

What college should a boy go to?

The boy thinking of geology solely as a pure (rather than as an applied) science can go to practically any college with a well-rounded scientific course. Not so the boys interested primarily in the engineering aspects of the mineral industry. They should enter a college offering specific engineering courses in mining, metallurgy, petroleum and the related professions.

The courses vary in length. Some of them run four years, others five. All of them are difficult. At least, they require serious and persistent work. Which is as it should be. Only a fully-prepared man can get ahead in the mineral industry today.

Is it expensive to secure an education?

It's not cheap. Tuition, books, board and other campus expenses should cost the average boy between $1,000 and $1,500 a year. However, this doesn't mean that a boy without that much money is barred. If he wishes, he can try for a scholarship. Most colleges give many. So do a number of industrial companies. And he can work his way through school. I did. I wish I had a dollar now for every furnace I stoked while I was at college.

Of one thing I'm sure. It's worth a struggle to get into this field. For the boy with ability and interest, who is willing to work, the future is unlimited.

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HOW TO HELP YOUR CHILD HAVE THE CAREER HE WANTS

Many factors will enter into your child's choice of a career: his interests, his ambitions, his abilities, the counsel he receives from teachers, friends and family. But, most of all, it will depend on his opportunities *to get the training he needs to enter the field of his choice.*

Even though his college days are still years away, it's never too soon to start making sure that your child will have the opportunity to continue his education when the time comes.

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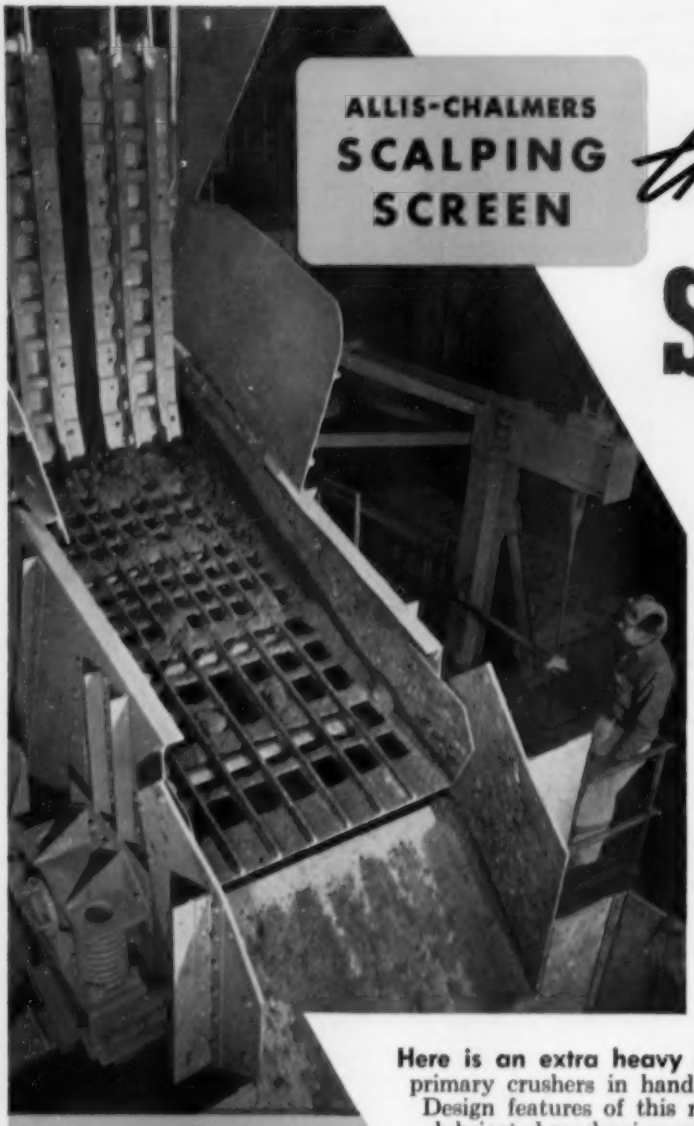

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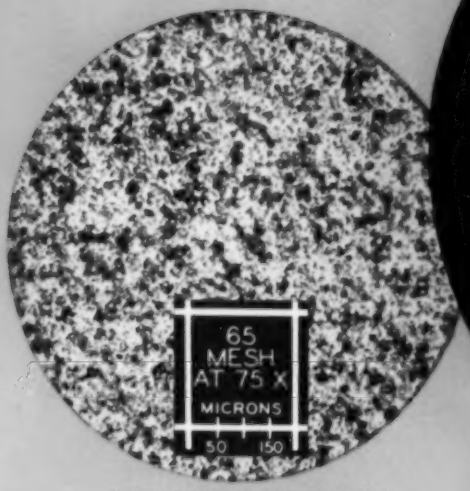

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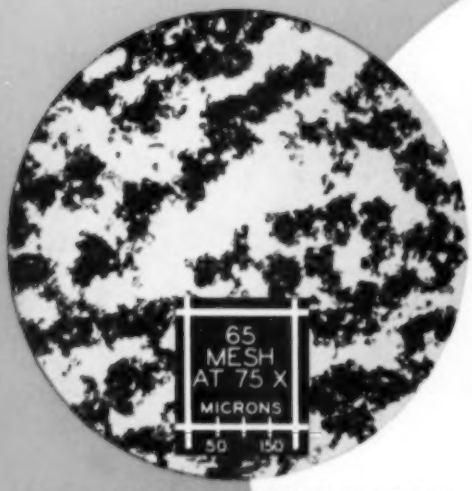

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Mine and Smelter has specialized in the design, manufacture and application of Marcy Mills since 1915. The extensive research and experience during this time has resulted in many refinements in the grinding process. One of the most significant developments is 2-stage grinding.

Experience has proved two successful ways of using Marcy Open End Rod Mills for 2-stage grinding...

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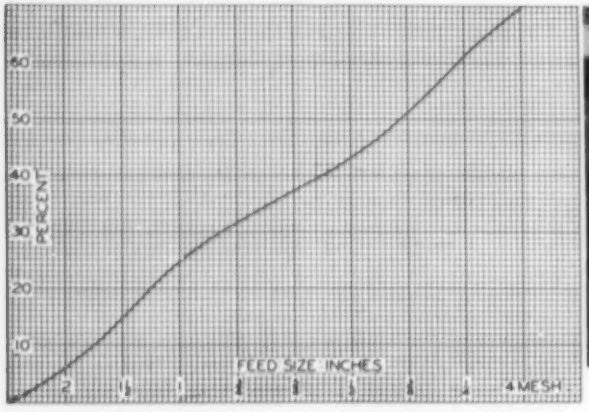
Operating experience has proved that use of Marcy Open End Rod Mills for 2-stage grinding will increase tonnage at lower overall cost per ton



This graph shows increase in mill capacity with decrease in size of feed. For example if a rod mill was used to decrease ball mill feed from 1" to 4 mesh the ball mill capacity would be increased 45%.

This large copper company installed a 9½' x 12' Marcy Open End Rod Mill in parallel with its 2-stage roll-screen circuit. As a result of this test 17 Marcy Rod Mills were installed to replace all roll-screen circuits. Rod Mill feed is 1½" hard ore; grind is to 4% + 10 mesh; tonnage up to 2700 tons per day, each mill. They reduced costs and improved operating conditions.

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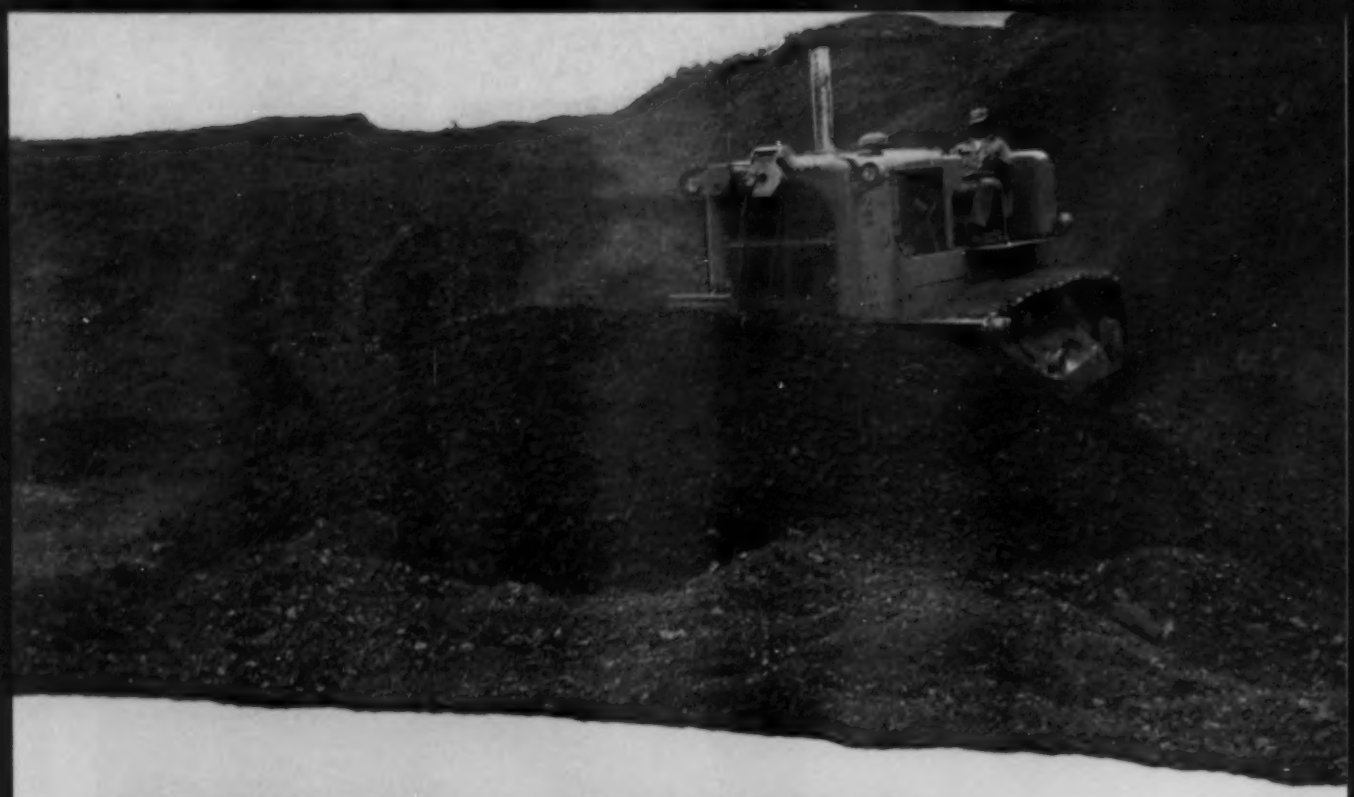

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Like all Cat-built equipment, the new D9s are engineered to do more work with less down time at low cost. Morrison-Knudsen Company, Inc., owns this and other giant Turbocharged 286 HP D9s here, purchased because of previous experience with Caterpillar equipment. The company also has Caterpillar D8 Tractors on this job.

The D9s are working in the phosphate beds of southern Idaho, largest known deposit in the world. The giant yellow "Kings of the Crawlers" strip overburden, which varies from 6 ft. to 10 ft. in depth, and feed ore to the shovel. These great new yellow machines are the product of many thousands of hours' testing on proving grounds and actual construction and mining jobs.

The D9 is tough and traditionally Caterpillar from its completely new, 4-cycle engine with Turbocharger to its "water quench" hardened track shoes. It's available with direct drive or torque converter for speeds to 7.8 MPH. Operators report it fast and easy to handle, with hydraulically boosted steering and brakes, "in-seat" starting, and smooth "live shaft" drive for cable controls.

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Steel Bolts in Mine Roof Support

by J. L. Humphrey

The origin of roof bolting is obscure, but is believed to have begun some 40 years ago in the mines of St. Joseph Lead Co. in southern Missouri. It was not until after World War II, and more particularly after 1949, as a result of the concentrated effort of the USBM, that roof bolting attracted widespread interest.

It is estimated that at the end of 1954, 26 pct of all bituminous coal mining in the U. S. was being done under roof bolts. Those mines alone use nearly 2,000,000 roof bolts per month, according to USBM statistics. Uses in metal mines are increasing rapidly, as confidence and knowledge grow. The application has found equal success in other fields, such as tunneling work for dam and waterway construction and right of way maintenance on highways and railroads.

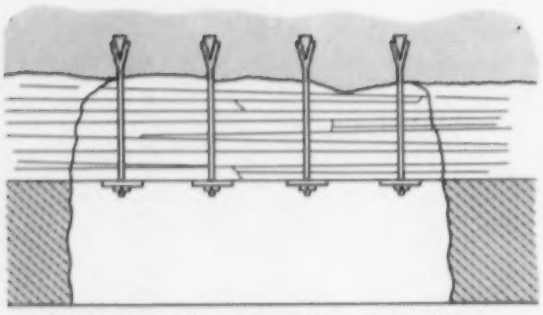
Analysis of USBM statistics for the fiscal year 1953-1954 reveals startling conclusions in regard to mine fatalities due to roof falls. On the basis of comparative accident frequencies, rating the number of accidents per million tons of coal mined, it would appear that 163 fatal accidents in bituminous coal mines during the fiscal year would have been avoided, if all bituminous coal mining had been converted to the use of roof bolts. The advantages in control of roof in hard coal and metal mining are equally impressive.

Steel bolts, properly installed, can provide an effective, safe, economic method of roof control in almost any underground opening.

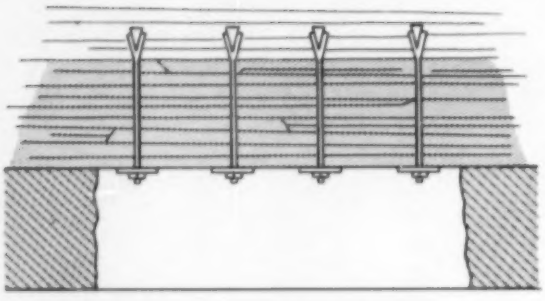
THE greatest number of fatalities in U. S. underground mines are caused by roof, face, and rib falls. Fifty-eight pct of the 370 deaths in bituminous coal mines in fiscal 1953-1954 were attributable to that cause alone. Yet frequency of roof fall fatalities in mines using steel roof bolts for primary roof control is one fifth that for mines using conventional timber support.* Because of that safety advantage

J. L. HUMPHREY, is Manager of Industrial Fastener Sales, Bethlehem Pacific Coast Steel Corp.

* U. S. Bureau of Mines reports, for fiscal 1953-1954, 0.257 fatal roof fall accidents per million tons of coal mined under roof bolts, 1.33 fatal roof fall accidents per million tons of coal mined where timbers alone are used for support.



Suspension of layers from firm strata above.



Bolting planes together to form a laminated beam.

and other features, extensive research has been done on roof and rock bolting by the mining industry, the U. S. Bureau of Mines, and steel manufacturers.

Theories as to support gained through bolting a roof vary with the type of mining and the ground to be held. In bedded structures the leading theories are:

1) The various layers of friable slate, coal, and shale are suspended by bolts driven into more substantial overlying strata. (Similar in principle to a suspension bridge.)

2) The various bedded planes are bolted together to form a laminated beam which prevents the roof from falling.

In unstratified formations two of the theories explaining the success of roof bolting are:

1) Any unsupported opening underground will gradually slough off rock until an arch contour is formed in which stresses are compressive. The spalling which occurs until a *caved arch* contour is reached results from tensile stresses in the back, or roof, which exceed the tensile strength of the rock (normally 2 to 10 pct of compressive strength). When roof bolts are used long enough to anchor in the natural arch, tensile stresses in the back are balanced and the rock will not slough off.

2) Each rock bolt in fragmented formation has the effect of squeezing together the rock fragments in a spherical zone around it so that a solid mass is formed. A ring of these rigid blocks around the perimeter of the opening acts as a masonry arch and the blocks support themselves.

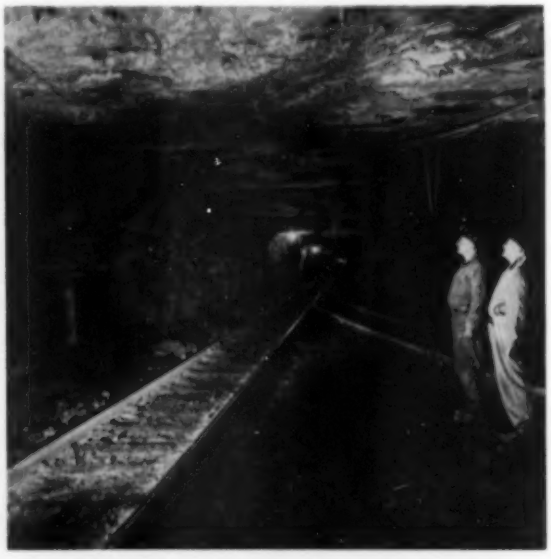
In all these theories the primary benefit of bolting is to supplement tensile properties of the natural rock structure. Bolting provides a reinforced rock lining with increased resistance to shearing movement due to the elastic properties of the steel reinforcing. In each case, when roof bolts are installed as soon as possible after the opening is made, the surface of the rock is put in compression, limiting the effect of elastic rock expansion by preventing the opening of cracks, joints, and cleavages that would be subject to chemical deterioration from air and moisture.

In practice, an underground opening that can be held with conventional timber or steel sets can usually be held with mine roof bolts, and in many cases bolts have been successfully used where conventional supporting methods failed.

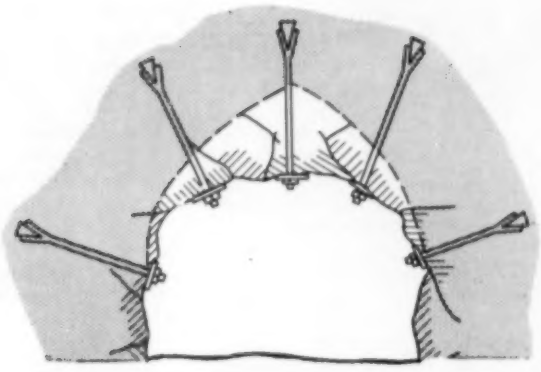
Roof Bolt Types: Roof bolts are manufactured commercially in several standard types. One common type is 1 in. in nominal diameter with a 1/16 to ⅛x-6-in. slot in the anchorage end and 5 in. of standard thread on the other. Bolts are anchored by means of a wedge driven in the slot during installation. The breaking strength of installed bolts ranges from 36,000 to 42,000 lb. Bolts are available in lengths from 2 to 10 ft, with threads produced by either roll threading or cut threading. Extensions and couplings are available for making up any desired length.

Another common type uses an expandable metal shell to provide anchorage. Normally ¾-in. diam mild steel bolts or ⅝-in diam high strength bolts are used. Bolts are threaded for 3¾ in. to 4⅜ in. on the anchor end to facilitate tightening the expansion shell. An unchamfered regular square bolt head is forged on the other end, with forging flash left on to provide additional bearing surface. The breaking load when anchorage is sufficient to develop the ultimate strength of the bolt is 22,000 to 28,000 lb. Bolts are available in lengths from 2 to 8 ft, with threads produced by either roll threading or cut threading. Special extensions and couplings are available to make up any desired length.

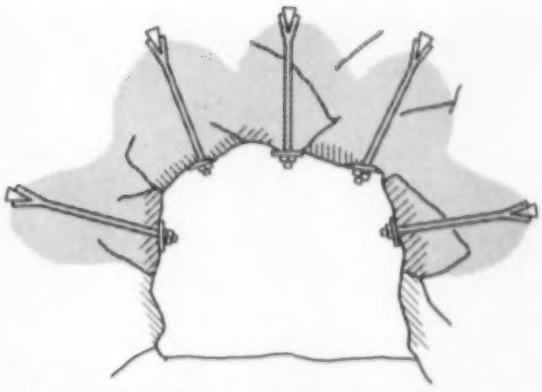
Many other types of supports have been tried in mines, particularly in the early stages. One mine in the Coeur d'Alenes is still employing a wooden pin, using wedges for tightening on both the anchorage and tunnel ends of the pin.



Slotted bolts used with square washers in roof of coal mine. The ideal bolt where maximum strength is desired.



Suspension of tensile zone from natural arch. Dash line indicates approximate contour of natural arch.



Area around each of the bolts squeezed together to form blocks acting as a masonry arch.

Regardless of the type of bolt used, bolting should be included in the regular mining cycle so that roof is always supported as soon as possible after it is exposed.

Two general types of expansion shells are available: self-supporting and those requiring forged ears on the bolt to hold the expandable portion of the shell in position. Self-supporting shells have a bale over the top which holds the expanding portion in place as the plug is pulled down in tightening.

To protect threads and to speed installation, many manufacturers ship bolts with shells assembled on the thread. Special keyhole and T-hole washers have been devised so that it is not necessary to remove the shell to place the plate washer on the bolt. The large hole of the keyhole washer permits the shell to pass through, while the T-hole plate is designed to be slipped over the headed end of the bolt after it has been partly inserted in the drilled hole. The latest development, however, is a plate with a 1⅜-in. hole. The bolt head is then made with a self-centering collar and a large head or a washer is used under the bolt head.

Bolting Patterns: Bolting patterns vary with rock conditions. A typical pattern in a 16-ft opening in a coal mine would be four bolts per row, spaced 4 to 5 ft in direction of the advance. The two center bolts would be driven vertically, or normal to the plane of the structure, while the outside bolts might be angled out about 30° to obtain anchorage over the rib. A steel angle washer is used to provide a bearing surface normal to the angled bolts.

In unstratified ground a pattern might consist of bolts driven in rows every 4 or 5 ft, one bolt vertically in the back; two bolts on each side, 4 ft from the center bolt, at a 15° angle; two additional bolts 4 ft down the side at a 30° angle. Special cast iron or steel beveled washers are used in that type of bolting to correct for angularity of the bolt, to bring the bearing plate parallel to the rock's surface.

In larger openings, such as motor rooms, intersections, and shaft stations, random pattern bolting is often employed. A typical pattern would find bolts installed on 5-ft centers in both directions. To provide further stability and to minimize deterioration from moisture and air, permanent installations may be gunnited after bolting.

Beams and Roof Ties: To increase the bearing area absorbing the load from the bolt and to control spalling between bolts, steel channels, usually 4 in. at 5.4 lb per foot or 5 in. at 6.7 lb per foot, or steel mine roof ties are used in place of mild steel plate washers. Wooden headers are used for the same purpose; however, gradual failure of the wood results in loss of bolt tension, destroying part of the effectiveness. Channels and roof ties are provided with holes punched for the spacing of bolts used in the particular formation. Holes are made large enough for either wedge or expansion shell to pass through. Special small plate and angle washers are used to distribute the load from the bolt to the channel or tie. In coal mines channels can be installed from rib to rib. In nonbedded structures they are more difficult to employ in long lengths. Roof ties, on the other hand, are flexible and can be adapted to surface irregularities of almost any ground condition.

Anchorage Capacity and Bolt Loading: The USBM has developed testing procedures to determine the length and type of bolt that should be employed and to measure the possibility of anchoring bolts in any given underground opening. From test results and experimentation, the spacing or pattern necessary to obtain the support desired is determined.

Measurement of anchorage capacity is obtained by pulling installed test bolts with a hydraulic pull tester. The tester is attached to the end of a bolt in the roof with the hydraulic jack bearing against the roof around the bolt, and load is applied until desired values are reached, or until slip occurs. Test results indicate the type and length of bolt that should be used to obtain maximum anchorage for the given rock conditions.

Danger of overloading the bolt with the impact wrench used in tightening during installation has led to establishment of safe limits of initial loading. Overloading may not be apparent, and an unsound roof may go unnoticed unless proper procedures are strictly adhered to.

USBM tests have established the ratio of tension in pounds to torque on the nut or bolt head:

For 1-in. slotted bolts, tension equals 42.5 torque-1000.[1]

For ¾-in headed bolts, tension equals 39.8 torque.[1]

Tests made by Bethlehem Steel Co. indicate wide variation and values in general lower than this unless hardened washers are used under the bolt heads, in which the case the tension-torque ratio is 60 for ¾-in. bolts and 80 for ⅝-in. bolts.

Recommended Roof Bolt Installation Procedure

Installation Sequence

Slotted type roof bolt:

1. Hole is drilled with stoper to depth 1½ to 3 in. shorter than bolt to be installed. Finish bit should be 1¼ in. diam.

2. Back of hole is sounded with rod or bolt to insure soundness.

3. Wedge is inserted into end of slot of bolt.

4. Bearing plate is slipped over threaded end of bolt and nut is screwed on thread so that about ½ in. of thread protrudes from nut.

5. Bolt with wedge inserted is started into the hole and threadless dolly fitted into threaded end of the bolt.

6. Bolt is driven to refusal with stoper. Driving bolt upward forces legs of slot over wedge and into surrounding rock, providing anchorage.

7. Nut is tightened with impact wrench to approximately 350 ft-lb torque.

8. Bolt is struck with rod or hammer. Experienced operator can tell by ring if bolt is tightened properly. Spot checks for tightness should later be made with torque wrench.

Wedges used with 1-in. bolts are usually ¾x⅞x½ or 6 in. long. Plates are normally 6x6 in. or 8x8x⅜ or 1½ in. thick, with hole centrally punched.

Headed type bolt with expansion shell:

1. Hole is drilled slightly longer than the bolt to be installed. Finish bit size should be 1⅜ in. for most expansion shells.

2. Bearing plate is assembled on bolt and expansion shell threaded on end of bolt.

3. Bolt is inserted in hole and turned to engage shell with sides of hole. As bolt is turned, plug of expansion shell is drawn downward, forcing the expandable sides outward into the rock, providing anchorage.

4. Bolt is tightened with impact wrench and tested by measuring torque with torque wrench.

For practical purposes a ratio of 40 to 1 may be used. Tests indicated that manufacturing methods, type of thread lubrication, type of bearing plates used, roof strata, and other factors appreciably affect the ratio.

For commercial 1-in. bolts the advised limit of installation torque is 350 ft-lb, or 14,000 lb. Tightening of ¾-in. headed or expansion shell bolts produces torsional (shear) stresses in addition to tension, lowering the tensile load at which yield may occur, and a limit of 200 ft-lb, or 8000 lb is advised.

The torque necessary to turn the nut or the head of the bolt can be read with a dial-type torque wrench. Provided that threads are free from rust and a straight pull is exerted, a true indication of load on the bolt can be obtained. Bolts should be spot checked after installation even though impact wrenches can be set for the desired driving torque.

Periodic spot checks should also be made in permanent drifts and accesses to detect any shifting of load. Unless the bolt is in tension, it is not providing adequate support. Loosened bolts should be retightened to insure that rock surface is held in compression.

Problems of Bolt Use: In addition to torque limitations, adherence to prescribed patterns and the size and depth of the hole drilled to install the bolt are of utmost importance. Successful bolting requires strict adherence to prescribed procedure:

1) A hole of accurate depth. A shallow hole does not allow sufficient takeup on the thread. A hole too deep does not allow proper wedging or tightening. For bolts using ⅜-in. or ½-in. plates, or washers, a hole 1½ to 2 in. shorter than the bolt is prescribed.

2) Tight, hard material at the back and sides of the hole. Since the wedge is driven against the back of the hole, it must be sound to provide good anchorage.

3) A hole 1¼-in. diam, finished size. A hole too large does not allow enough bite for the legs of the slot when they are wedged out. Thicker wedges may be used if holes are oversize.

Diameter of the hole drilled and protection of bolt threads and expansion shell are of primary importance for proper anchorage of the headed-type bolt. Protection is afforded by shipping and storing bolts with the expansion shell assembled on the bolt.

Very good results are obtained where the hole at anchorage depth is of the prescribed size, usually 1⅜ in. A larger hole reduces both anchorage capacity and effectiveness of the expansion shell. Most shell types are sufficiently strong to break a commercial steel bolt if the hole is drilled to the proper diameter and the rock structure is firm.

Metallurgical and Manufacturing Practices: Since the ratio of initial loading to ultimate tensile strength is low, steel manufactured for mine roof bolt application requires close adherence to specifications to insure sound and uniform product. Commercial steel bolts are made from new billet steel to meet Spec. A-306, latest issue, American Society of Testing Materials. Grades 60 or 65 are for regular strength and grade 80 for high strength bolts.

To meet that specification and to supply proper steel for end use requires special control of the raw materials making up the charge to the open hearth furnaces.

Deoxidation practice is directed toward producing homogeneous structure. Killed steel is normally applied to provide uniformity throughout the cross-section of the rod. Special discard is usually neces-

Slotted bolts with square washers are shown. Headed type of bolts with shells can be used in the same manner.

sary in billet rolling practice to insure removal of segregation and pipe.

Bars for roll-threaded mine roof bolts must be produced to closer than standard mill tolerance, for the rod diameter must be within the pitch diameter tolerance for the thread to be produced in order to insure uniform thread. Roll threading, the formation of a thread by rolling the steel through grooved dies that imprint the thread form in the steel, increases the physical properties of the steel by cold-working the metal. The resultant thread is stronger than a cut thread of equal size and offers improved thread surface properties. As the threaded portion of mine roof bolts is normally the point of smallest cross-section, a roll-threaded bolt offers equal, or greater, strength with decreased weight of steel.

The slotting operation must also be closely controlled to maintain maximum cross-section in the rod. Burned slots should be kept to minimum width (⅛ in. or less) and burner adjustment must be optimum to blow all slag from the cut. With careful control of torches and gas, and ideal cutting conditions, curfs 1/16 to 3/32 in. thick can be produced on 1-in. rods. Forged slots are advantageous, since no metal is removed in the slotting operation and the slot is automatically centered.

Assembly of shells or nuts on the bolts prior to shipment not only offers savings in installation time, but insures proper fit between the two parts. Final inspection for defective product is accomplished when bolts are bundled for shipment.

Advantages of Roof Bolting

Use of roof bolts instead of conventional timber sets offers many advantages, both from the safety and cost standpoint:

1. Safer, more assured roof control. By systematic bolting before any sag has taken place and by providing sufficient tension in the bolts, roof sag can be limited and roof falls almost eliminated.

2. Lower handling and timbering costs. Reduction of timber in the mine reduces fire hazard.

3. Less storage space required.

4. Cleaner ore and coal through better control of sloughing of waste rock.

5. Reduced danger to men from falls caused by blasting and accidental dislodging of timber posts. Danger from falls in event of train wrecks also lessened.

6. Increased head room and side clearance due to absence of timbers, resulting in increased productivity.

7. Availability of bolts for hanging air, water, and messenger lines.

8. Increased height of working face. Timber sets limit roof heights to approximately 14 ft before buckling of the legs becomes a problem. Roof bolts allow increased height and, therefore, increased recovery of coal or ore. (One Utah coal mine is now mining the full depth of a seam 25 to 30 ft thick, removing 80 pct of the coal. Previous recovery limited to only 14 ft of the seam depth was estimated at 40 to 45 pct.)

9. More permanent support. Bolts are estimated to last 20 to 40 years, depending on corrosive conditions.

10. Reduced sloughing of rock caused by attack of air and water. Fractures and faults are kept from opening.

11. Decreased excavation costs. Openings need not be made larger than working space required. In one Butte mine of the Anaconda Co., 2500 ft of recent development was accomplished with 37 pct less excavation than would normally be required. Handling 10,000 tons of broken material was avoided through the use of roof bolts.[5]

12. Decreased ventilation costs through reduced resistance to air flow. One mine reports that a single ventilation unit now handles two levels, where one unit per level was required with conventional timbering. Estimated cost per ventilation unit—$70,000.

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Platinum Mining In Colombia, South America

by Patrick H. O'Neill

THE Choco placers, mined continuously since the earliest expeditions of the Spanish conquistadors, were exploited by the Indians prior to the conquest. Platinum is found in varying percentages with gold throughout the Choco placers and is found in minor percentages in some placers in other parts of Colombia. The Choco Department is on the west coast of Colombia between the western range of the Andes and the Pacific Ocean and extends from the San Juan River north beyond Panama to the Caribbean.

The first reference in European literature to platinum was by Julius Caesar Scaliger in 1557. He did not assign any name to it but referred to it as an infusible metal found in the mines in the New World. The Spaniards had given it the name *platina*, which is the diminutive for *plata*, the Spanish word for silver. Don Antonio de Ulloa first recorded the name while on an official visit to the Choco from Spain in 1735.

Although the Spaniards, at that time, considered the metal infusible and worthless, there is ample evidence that the pre-Inca Indians manufactured platinum jewelry. Many ornaments and pieces of jewelry that are a crude alloy of platinum and gold have been found in old tombs near Esmeraldas in the northern part of Ecuador. In a report by Paul Bergsoe of Copenhagen on the metallurgy and technology of gold and platinum among the pre-Colombian Indians, he arrived at the conclusion that the Indians did not have the means to fuse platinum but that by mixing grains of platinum with gold dust and heating with a blow pipe on charcoal, the fused gold would permeate the platinum and simultaneously the platinum would dissolve to some extent in the molten gold. Repeated forging and heating would gradually produce an homogeneous mixture which would then be worked into the desired shape of the ornament or jewelry. Had the Spaniards known the Indian method, the entire history of platinum would undoubtedly have been much different.

The properties of platinum were first discovered in 1750, and a French chemist, Chabaneau, discovered a commercial process for working platinum in 1786. It was not until 1838, when Robert Hare developed the oxy-hydrogen blowpipe, that platinum really came into its own for commercial uses in the chemical industry. Discovered in Russia about 1822, platinum was used for coinage until 1845, when the coins were worth more than their face value. Russia was the largest producer of platinum up until the time of the Revolution prior to World War I, when production was sharply curtailed. Colombia then became the principal world producer from 1916 to 1923.

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When first encountered in Colombia, platinum was considered by many to be unripened gold. Reports are that when it was first sent to Spain, it was returned to be put back in the ground until it ripened.

Colombia was the first country to exploit platinum deposits and has been a consistent producer, especially so since dredging began in 1915. The annual production by modern mechanical dredges has seldom, however, exceeded that of the native miners, who are today mining with the same hand methods used since the Spanish Conquest.

Platinum has slightly higher specific gravity than gold and occurs in alluvial deposits in small grains and flakes, similar to the gold, so it is difficult to separate the two metals. Amalgamating the gold with mercury facilitates the separation, but the individual miners generally could not afford the mercury. It has been reported that mercury was used in the Choco as early as 1720 to amalgamate the gold and that for a time it was required to assist in stopping platinum counterfeiting in gold bars.

With the platinum practically valueless and so close in weight to gold, there were many attempts to use it for counterfeiting. Coins and bars made of platinum were hard to detect by gravimetrical methods, and a thin coating of gold, or gilding, gave the article the appearance of gold. The Spanish doubloons, which were international currency at the time, were frequently counterfeited in platinum. The encyclopedia of Diderot and d'Almbert, published in 1744, tells of Holland sailors receiving counterfeit coins on the coast of South America and goes on to say that they stopped over on their next voyage and hanged the traders who had given them the coins.

In 1778 Charles III of Spain ordered that all platinum produced in his American possessions be turned into the Royal Treasury. No payment was made for the metal and traffic in it was forbidden, so the metal was thrown away as it was separated from the gold at or near the mines.

The miners generally rough-panned the dirt at the working area and placed the concentrates from each pan in a gourd which they cleaned later. They either did the cleaning at their houses and threw the platinum and black sand out, or did the cleaning close to the river and threw the platinum back in.

After platinum became valuable, and especially during World War I when the price was very high, the local inhabitants started re-panning all the areas where they knew or suspected that platinum had been thrown out in previous years. In the town of Quibdo, the capital of the Choco Department, it has been told that the municipal authorities claimed all material in the streets for the municipality and directed the carrying of the material to the river where it was washed, the platinum recovered, and the tailings brought back to fill in the streets. All the inhabitants in the town dug up the areas under and around their houses, and in one instance a large house was torn down to wash the material in the bricks and under the house. Enough platinum was reportedly recovered for the owner to rebuild his house and still have a $4000 profit. Many people searched for the sites of earlier slave mining camps to find the places where it had been thrown away, and a few lucky ones were well rewarded for their search. Occasionally the dredges have had unusually high percentages of platinum in a weekly clean-up, which also substantiates these stories, as the ratio between platinum and gold is normally very constant over large areas.

At the beginning of World War I, the price of platinum rose to a value slightly higher than that of gold. As it became scarcer, due to the drop in Russian production, the price soon exceeded $100 an ounce. This added considerable impetus to the plans for large-scale production that were already well under way in Colombia.

The main platinum area in the Choco was the Condoto River, a tributary of the San Juan River. Exploitation of the Condoto River was handicapped by title disputes that arose from a concession grant on areas already denounced for mining titles.

How It Happened

In 1907 General Ciceron Castillo, who had been an important general during the War of One Thousand Days, was granted a concession for mining the Condoto River by General Rafael Reyes, the President of the Republic. Castillo formed a company for this purpose in Bogota, took a prospect drill into the property and opened a small hydraulic operation. His operation was not very successful, so he asked General Reyes for permission to sell his concession to a foreign firm. General Reyes agreed, but with the understanding that such capital should be European.

Castillo promoted the property to Consolidated Gold Fields in England, and the Anglo-Colombian Development Co. was formed to buy the Castillo interest. In 1911 and 1912 the area was drilled and shafted and other properties were acquired. The camp was established in 1912 at Andagoya, dredge parts were brought from England in 1913, and the dredge started digging in July 1915 in the Condoto River. Then the disputes over titles began because other individuals also claimed title to the Condoto River through the denouncement of mines prior to the time the concession was granted to Castillo. One was Henry Granger, who had been in Colombia since 1889 exploring and prospecting the rivers of the Atrato and San Juan drainages. Granger had obtained title to large areas that he considered dredgeable. One of these areas was the Condoto River, from the mouth up to an island known as Bazan. Two local men, Antonio Asprilla and Jose Mayola, had denounced mines from the Island of Bazan upstream.

Henry Granger, backed by the Lewisohn interests of New York, formed the Pacific Metals Co. After some suspension of operations by court order, the Pacific Metals and Anglo-Colombian companies settled their dispute and South American Gold & Platinum Co. was organized in the U. S. as a holding company with a subsidiary, Compania Minera Choco Pacifico, as an operating company in Colombia.

About this same time the British Platinum & Gold Corp., formed in London, brought a small dredge to the upper Opogodo River, a short distance away from the Condoto operations. Since the area between the Condoto and the Opogodo was known to be rich in platinum, the two companies immediately started haggling over the properties in that area. Both companies had applied for titles to certain areas at the same time. In order to keep possession, they each had miners working by hand in the area. A court decision was finally made in favor of South American Gold & Platinum Co.

At the same time, the British Platinum & Gold Corp. made a deal with Asprilla and Mayola for their mines above the Island of Bazan in the Condoto River, and a legal dispute started over that area again. The Supreme Court, however, handed down a

decision recognizing the Castillo concession. The British Platinum & Gold Corp. then had no place to work their second dredge, which had just arrived, except in a small area on the side of the river that was excluded from a large area controlled by South American Gold & Platinum Co. The dredge dug only a few months when it accidentally turned over and sank, and it has lain there half submerged ever since.

The first dredge of the British Platinum & Gold Corp. dug for several years in the small area that they had near Opogodo. Production was never very large, and although the dredge was in good ground, the vibrating screen that it used was not very successful, recoveries were very poor, and the company ceased operations in 1927.

Meanwhile, South American Gold & Platinum Co. was expanding. It placed a second dredge in operation in 1920, a third in 1923, and a fourth dredge in 1932. A fifth dredge was placed in operation under a separate operating company farther south in 1937, and a sixth dredge was added in 1938. The original dredge built in England was steam-driven with an open connected bucket line. It was later rebuilt and converted to a more modern style dredge with close connected buckets and standard riffled sluices, and it was entirely electrified. This dredge has been operating steadily and after 40 years is still on the original hull. The hull is at present very thin, however, and a false bottom will be installed before the dredge is moved to another property next year.

The second dredge was of American design, similar to the California dredges of that time, and was diesel electric. This was also assembled in Colombia. It made good money for quite some time re-digging the tailings of the first dredge. The third dredge was constructed in Pittsburgh and towed down the Ohio and Mississippi Rivers, across the Gulf of Mexico, through the Panama Canal and up the San Juan River to the Condoto River. It was also a California-style electric dredge. The dredge was equipped with a boiler that could be either wood-fired or oil-fired to drive the generators. Wood was first used, but it was difficult to dry the wood in the rain-soaked jungle, so oil was generally used. More than 100 natives worked steadily with their own canoes hauling the oil upriver three to five barrels at a time. They were paid on delivery at the dredge and there was no recourse for loads that were lost.

One of the many difficulties encountered in the early years of the company was obtaining enough silver coins to pay the natives. The only money they would accept was *plata vieja*, or old silver that was a combination of coins from Peru, Mexico, and other places, including some shillings. The natives would not accept any paper money because they buried their money for safekeeping. The *plata vieja* was being removed from circulation and it was necessary for the company to open a store in Cali to obtain the silver to pay the natives until they learned to accept the new Colombian currency.

A hydroelectric plant was built between 1922 and 1925 about 30 miles north of the operations on a tributary of the Atrato River. The plant is a lowhead plant built where the Andagueda River doubles back upon itself leaving only 325 ft between the legs of the *U* with a 14-ft difference in elevation. The 2000 kw produced by the hydroelectric plant was later augmented by a diesel plant at Andagoya.

All the dredges have been rebuilt and modified several times in an effort to meet rising costs with increased capacity and improved performance. For example, the second dredge originally had 5-cu ft buckets and dug at 22 buckets per min. The bucket size has since been increased to 7 cu ft and the speed increased to 34 buckets per min. Instead of the 100,000 cu yd per month for which the dredge was designed, it now averages close to 300,000 cu yd per month. Recovery facilities have not always kept pace with the increased yardage but the table area has been increased, larger pumps have been installed, and the screens and stackers have been speeded up. Similar changes have been made in all the dredges.

Losses have undoubtedly increased to some extent during the increases in yardage handled, but with the greater capacity it has been possible to dredge ground previously too low in tenor. Recent testing with a jig unit has shown that most of the values being lost can be recovered by jigs and one of the dredges is to be converted from tables to jigs very soon. Other dredges will be converted later if the first modification comes up to expectations.

The percentage of platinum to gold varies from 75 pct platinum by weight of total metal in the Condoto River to as low as 1 pct platinum in other areas. The gold production by the company is larger than the platinum in total ounces but generally about equal in value, due to the higher price of platinum.

Designed for digging in the swift rivers, all the dredges have tapered bows and sterns, with high leads for the bow and stern lines to clear the passing river traffic. They are also equipped only with head lines for river dredging. A spud was added to two of them for inland pond dredging and a third dredge will soon be modified.

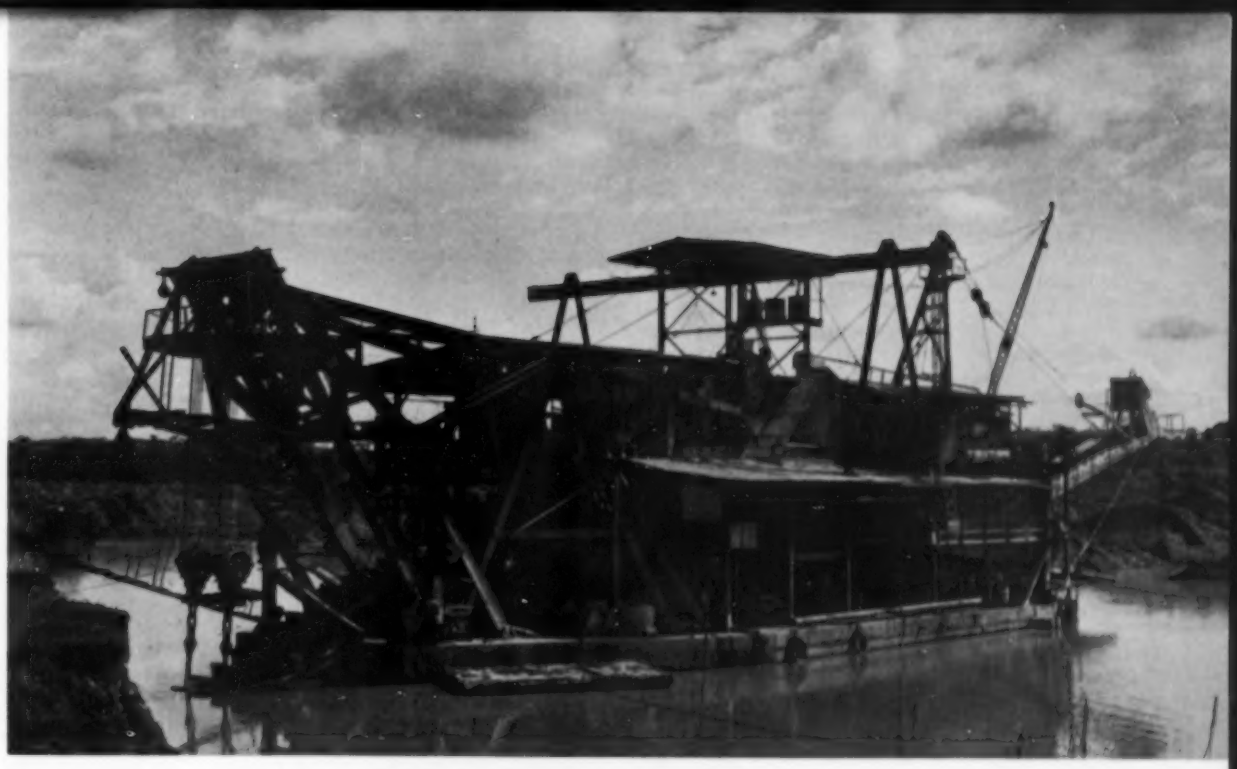
The dredges are, for the most part, digging in inland ponds now, as the river areas with dredgeable values have mostly been completed. Dredging on inland ponds involves considerable extra expense, as the surface rights in Colombia are not included with the mineral rights and the surface rights have to be bought or leased separately. The surface is either virgin jungle that must be cleared by hand, as it is too swampy to be worked on with any equipment, or it is planted with bananas, platanos, sugar cane or some other crop and the crops have to be purchased and cleared.

Hand Work Continues

Native miners work terrace or bench deposits along the rivers. In working the terrace deposits, they dig a ditch along the base of a gravel bench or terrace to the false bedrock. They then dig into the material with iron bars or steel tapered poles, dumping the dirt into the ditch where the material is later washed by rain water. Water is no problem in this area, as rainfall averages 300 in. per year.

Some of the miners make small reservoirs to impound water for later sluicing, but most of them let the rain run-off do the sluicing during the night. There is generally enough water the following day for cleaning the rocks and panning. Rocks and tailings are thrown to the opposite side of the ditch after the lighter material is washed away, so the ditch continues to advance into the face of the workings. After several days of sluicing, the bottom of the sluice is concentrated by raking with a short stick that has hook metal blades on the end. The final material is then hand-panned in bateas.

Native miners are permitted to work anywhere in navigable rivers regardless of mining titles. The only restriction is that they are not supposed to work within 100 meters of an operating dredge. Their



ABOVE: Dredge No. 6 working the main platinum area, Big Flat, near the San Juan River. With 10½-cu ft bucket capacity the dredge averages 13,000 cu yd per day. RIGHT: Big Flat area being worked by dredge No. 6. Tailing piles which show as ripple marks will tend to level off and within five years will be green, turning back to jungle area.

ABOVE: Lowhead hydroelectric plant at La Vulta, Colombia, with capacity of 2000 kw, supplies the main power for dredges. Arrows indicate flow of water in the Andrageda River. LEFT: Dredge No. 2, with 7-cu ft buckets, maintains a 42-ft face below the water and 10-ft face above. Natives can be seen cleaning away tree roots and other debris, which hamper operation.

work in the rivers is carried on by several different methods. Since the rivers are frequently in flood stage and a 12 to 24-ft rise is common, considerable re-concentrating takes place on the upstream sides of the bars and behind large boulders, so they work on the upstream sides of the bars when the rivers are low. They also work in the rivers in low stages, frequently diving to fill their bateas with dirt from the bottom of the river. In fairly shallow streams, they tie a rock to their backs to help them stay on the bottom while they fill their bateas. In rivers where the water is over their heads, they use a larger rock and release it under water so they can get back up. They frequently use a notched log on which to climb back out with a heavily loaded batea. Many of the divers are women, and it is said that it is possible for them to stay under water up to two minutes and sometimes longer.

Frequently the native miners build wing dams out from the bank to slow the current in shallow, swifter rivers so that they can dive and not become too tired fighting the current. In some places they build dams across the river and open up small sections at a time to sluice off the barren material so that they can get down to better values.

Although the natives are restricted by law from working close to the dredges, whenever a dredge is in particularly good values, the news travels quickly. The miners then become quite a problem, as they start working in the face or side of the dredge cut and frequently dive down to scoop material off the buckets as they are coming up. It is very dangerous for them to work around the dredge and the operation is handicapped until the proper legal proceedings are complied with to get the authorities to order them away.

No accurate figures on the number of native miners are available, but it is estimated that somewhere between 20,000 to 25,000 are working. From public production and export figures, it is apparent that the miners produce approximately as much platinum as the dredges. Production and export figures are not always accurate because there is some smuggling. This was especially so during the two World Wars, when Germany needed platinum and had agents in Colombia buying from the individual miners. Effort was made, of course, by the Allies to prevent the platinum from going to the Germans, and it was bought by Allied agents from the miners during both wars and exported through legal channels. During the last war, one Allied agent alone was known to have bought approximately $300,000 worth of platinum from native miners.

Native miners, of course, must limit their operation to areas of high values in platinum and gold. The average value of the material the dredges handle would not be worth the time of a man working by hand, since profit depends on mass production. Also the dredges recover the metals that are buried too deeply for hand mining and the bench deposits are purposely left to the native miners, even though many of these deposits are included in titles that are vested in the company. It is often difficult for local people to realize the advantages of an industry in their area.

South American Gold & Platinum Co., instrumental in developing the platinum industry of Colombia, has been the largest single producer of platinum in Colombia for nearly 40 years. The dredging operations, the only large industry in the entire Choco, are very important to the economy of the area. The operating companies have established schools and medical facilities and have done much to improve the health and literacy of the local inhabitants.

The success of the operations under difficult operating conditions has depended, to great extent, on individuals who directed the activities. The principal among these was Newton C. Marshall, an American mining engineer who worked a year in Panama and prospected in Colombia before taking over as resident manager at the time the Anglo-Colombian Development Co. and Pacific Metals Co. joined forces. Mr. Marshall continued in active charge until his retirement a year ago. He was greatly aided throughout all the legal difficulties in Colombia by H. H. Standing, an Englishman who acted as British Consul in Panama in 1903 and later prospected and mined and worked for various firms in Colombia prior to starting work for the company in 1917. Mr. Standing is still in the employ of the company, and although his activities are somewhat restricted by failing health, he is still an important adviser and consultant on mining titles and other legal matters in Colombia.

There have been many improvements in living conditions and transportation during the years the company has operated, but the jungle is still dense and the rivers treacherous, so the problems of operating and moving the dredges are as great as ever.

Rate of Production

Although annual production of platinum from Colombia has remained fairly constant since 1917, its percentage of the world production has dropped from 50 pct or more between 1916 and 1923 to less than 5 pct in recent years. The world producers, in order of production, are South Africa, Canada, Russia (estimated figures only), Colombia and the U. S.

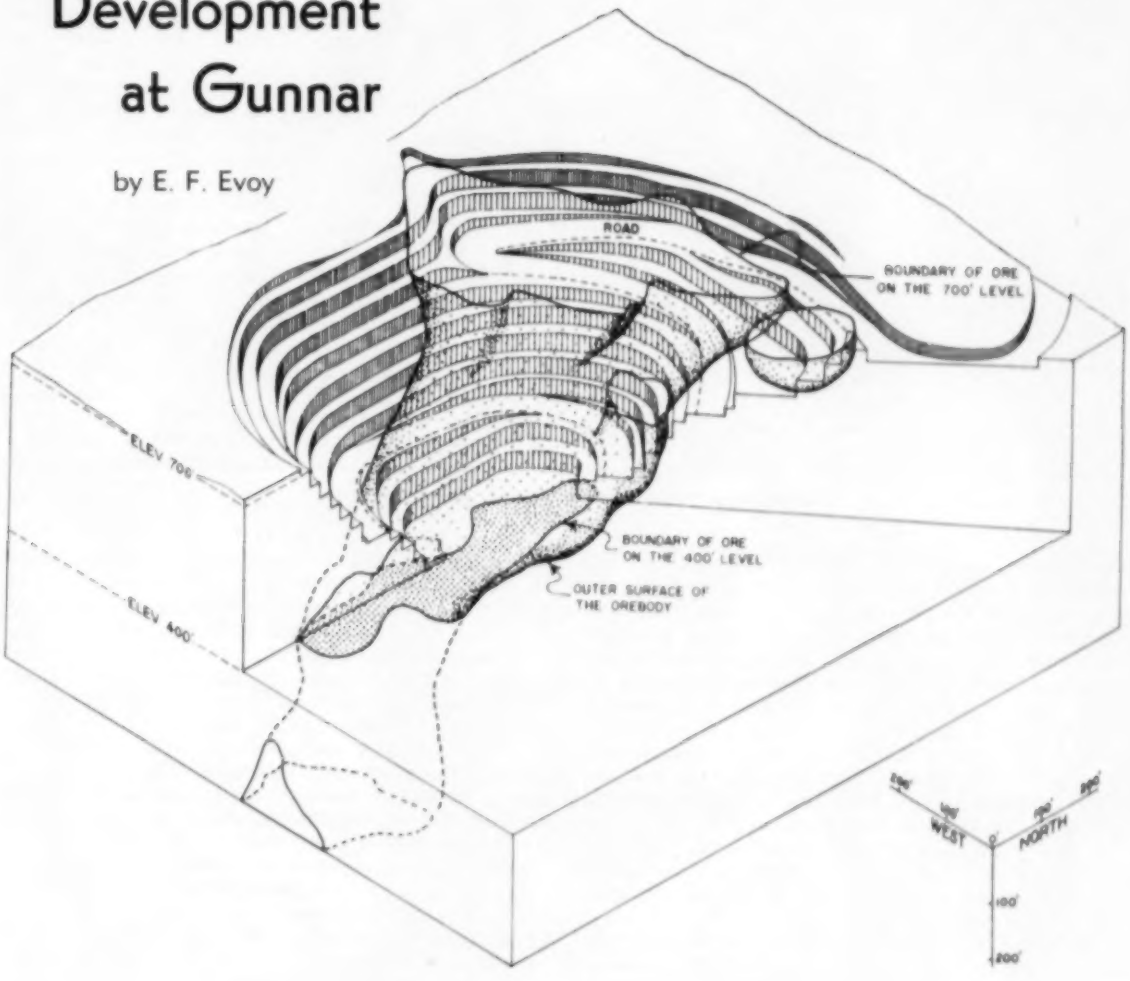
The dredging industry throughout the world has been hit by rising costs just as other industries have. However, these increases have been met and overcome to some extent by technical developments and improvements that have been continuous since the first successful bucket lift placer dredges began operating in California in 1898. The placer dredge is today one of the most highly developed and efficient pieces of mining equipment. Many modern dredges handle 13,000 to 16,000 cu yd per day. These volumes, converted to terms generally used in the mining industry, are approximately 23,000 to 26,000 tpd, tremendous tonnages for a floating unit to dig and process every 24 hr.

Improvements in efficiency and capacity have made possible the mining of lower grade reserves, since operating cost per dredge has been held down through improved efficiency, so that the additional production capacity has resulted in reduced unit costs. Improved maintenance techniques have offset, to a large extent, rising material and labor costs. Electric welding has been the most important factor in reducing maintenance costs on dredges by prolonging the wear-life of large expensive items.

A change in management of South American Gold & Platinum Co. occurred in July 1954. Since that time the company has diversified its activities and is interested in developing other natural resources both in Colombia and in other South American countries. The company has had many years of successful operation in Colombia and continues optimistic about the future of private investment both in Colombia and in other Latin American countries.

Open Pit Development at Gunnar

by E. F. Evoy



Open pit operation goes ahead despite average winter temperature of —10°F.

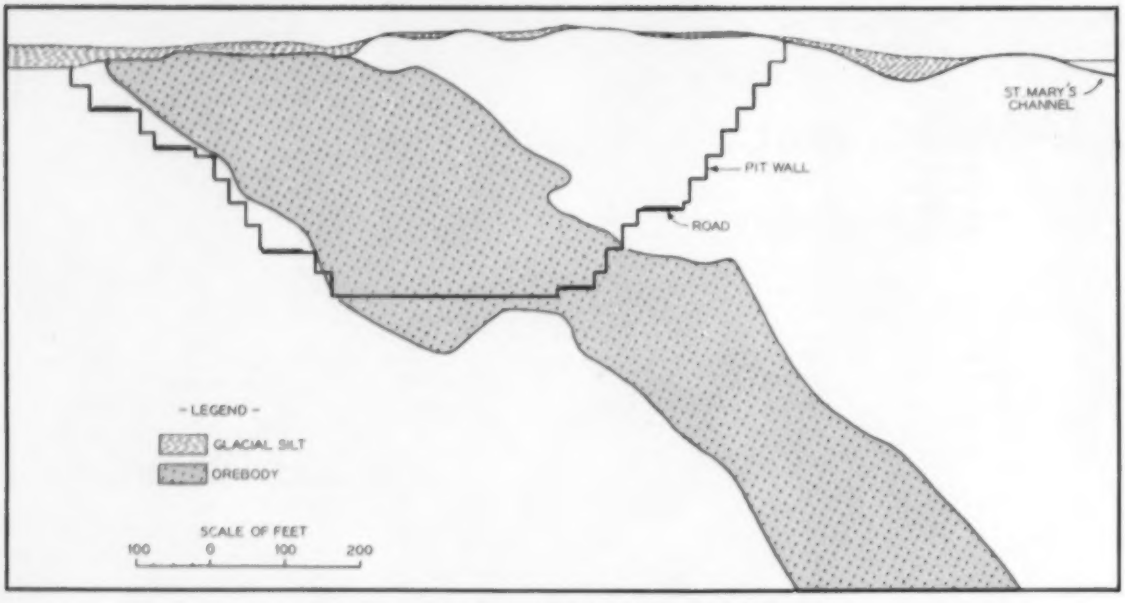
THE uranium deposit of Gunnar Mines Ltd. is on south shore of Crackingstone Peninsula in the Beaverlodge area of northern Saskatchewan. The property spans St. Mary's Channel, a narrow waterway separating the mainland from a group of islands in Lake Athabasca.

The mine property, consisting of 12 claims and 7 fractions, was staked in July 1952 by A. O. Zeemel and W. J. Blair. Diamond drilling commenced in the fall of that year and continued until the spring of 1954. In that interval, some 70,000 ft of drilling outlined an orebody plunging from the surface southward to a depth of 1000 ft below the shore of St. Mary's Channel. The last 9000 ft of drilling was through lake ice, and when this was interrupted by spring breakup, sufficient ore reserves had been established to warrant the present operation. Surface drilling was therefore terminated.

E. F. EVOY is Mine Geologist for Gunnar Mines Ltd., Uranium City, Sask.

The area is underlain by an assemblage of Pre-Cambrian metasediments striking N 75° E and dipping at 45° to the south. The lowermost rock is a fine grained quartz-feldspar paragneiss occupying the northern and western portion of the claim group. This is overlain to the south and east by a coarse grained well foliated granite gneiss. Near the contact of the two, but within the granite, is an irregular body of syenite that differs from the granite gneiss only in the presence of calcite and the corresponding absence of quartz. The northeast corner of the syenite mass is brecciated and mineralized.

The ore consists of disseminated pitchblende and secondary uranium oxides in the brecciated syenite.



Longitudinal section of Gunnar orebody and open pit, looking east.

Associated gangue minerals are virtually absent, although much of the ore and surrounding rock is stained red with sooty hematite. As a consequence of the hematite discoloration, the color of the ore may vary from pink or gray to brick red. Similarly the texture may vary from coarse grained granitic to finely granular, depending on the degree of brecciation of the syenite. The granular ore is usually porous, the granitic variety frequently vuggy, due largely, it is believed, to partial leaching of the calcite. However, none of these features, color, texture nor calcite content, is sufficiently diagnostic of the ore. Since the pitchblende is commonly invisible megascopically, it is seldom possible to distinguish between ore and unmineralized syenite except with the aid of a radioactivity detector.

Orebody: The orebody is an irregular pipe-shaped mass outcropping at the surface and plunging southward at 45° for a known length of 1400 ft. It has a maximum diameter near the surface of about 450 ft. Between depths of 250 and 350 ft, the horizontal configuration changes from roughly circular to figure eight shaped. The north circle of the 8 becomes smaller and disappears; the south part enlarges and continues the southward plunge. The effect is that of a bend or offset in the pipe. Whether this feature is accomplished by a change in plunge or in a slight vertical displacement along a fault is not known. Nevertheless, it creates a convenient mining arrangement. The northern, diminishing portion of the 8 forms the bottom of the open pit; the remaining ore will be removed by underground mining.

Although geological conditions are favorable to open pit mining, the topography is considerably less so. The mine area is one of low relief with bedrock elevation ranging from 650 to 750 ft (datum level: Lake Athabasca 700 ft). The bedrock surface of the ore was initially overlain by glacial silt and muskeg. The pit at surface approaches to within 100 ft of Lake Athabasca and will lie almost wholly beneath lake level. However, of the 180 holes drilled, only a few of the deep ones made water, and analysis indicated the water to be markedly different in composition from that of Lake Athabasca. The minor faults encountered were for the most part gouge-filled and dry. Management is thus encouraged in the hope that no serious water problems will be encountered.

Plan of the Open Pit: The plan of the open pit is shown on the accompanying diagram. At elevation 700 ft it is 975 ft long in a north-south direction and 775 ft wide. Ultimate depth will be 300 ft. Except on the west side, the upper walls are in granite gneiss. The north wall follows closely the footwall of the ore so that the waste is largely confined to a crescent-shaped mass in the south end, diminishing quickly with depth. The waste to ore ratio is 2½:1. The rock will be removed in ten 30-ft benches, leaving a 21-ft berm and 30-ft vertical face on each bench for an ultimate wall angle of about 55°.

Climbing in a spiral up the walls, with only one serious turn within the pit, the road is 50 ft wide. Total length, from bottom of the pit to the mill slip, is slightly less than 1 mile. Grade never exceeds 8 pct. The three passes on the north end of the pit reduce the standing angle of the north wall which, being on the dip slope of the foliation in the granite gneiss, is theoretically the weakest wall.

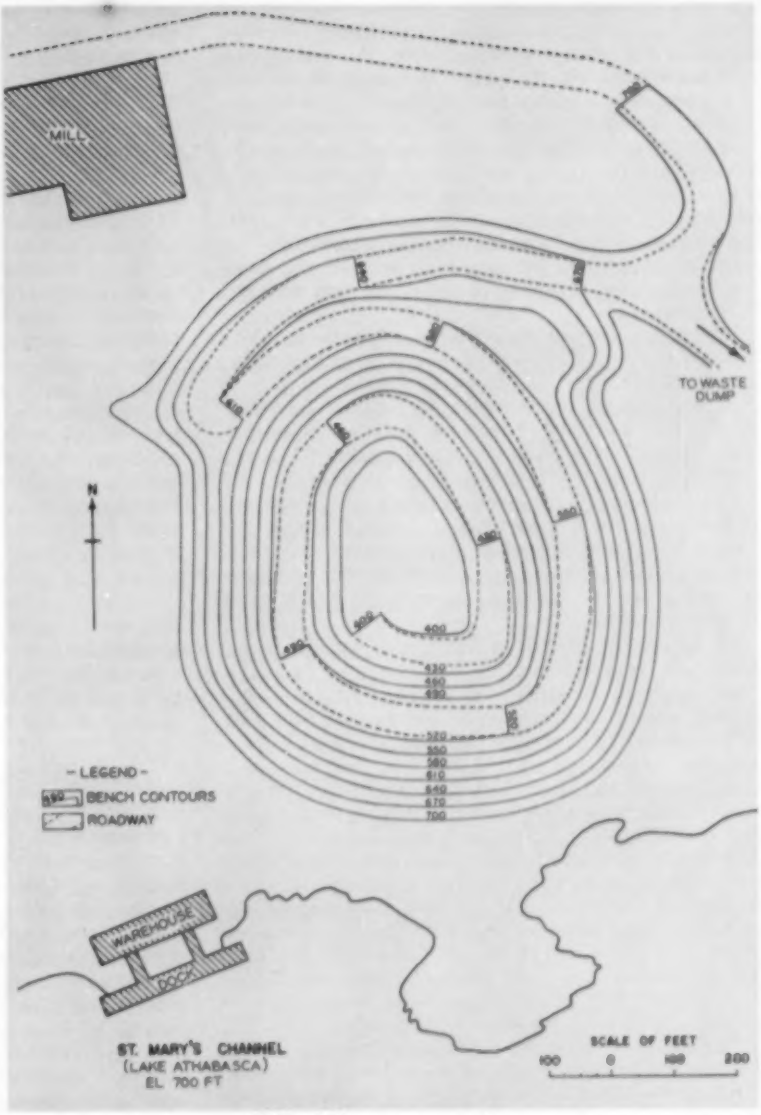
Overburden Removal: Perhaps the most difficult problem encountered in the pit development, and to a lesser extent in road and plant construction, has been removal of the glacial silt overlying the bedrock. The silt *in situ* is brown, damp, and thinly bedded, very similar in appearance to varved clay. Like clay it is cohesive, almost impervious to water, and drains slowly and poorly. Subjected to traffic, however, it deteriorates quickly to form a soupy mud. If used for road fill, it caves and produces frost boils; on the rock damp it causes slumping; and in the mill circuit it slimes in the thickeners. When dry it crumbles quickly to form a powdery, abrasive dust. The terms *rock flour* or *glacial flour* aptly describe it.

Over the north end of the pit the silt had a maximum depth of 60 ft and an average depth of 20 ft. This was capped by 2 to 3 ft of muskeg and light brush. Beneath the muskeg the silt was frozen

and interbedded with multiple lenses of ice ranging from less than 1 in. to 3 ft thick. Frost action was commonly confined to the upper 5 ft of the silt, but locally reached depths of 20 ft. Since this existed under a mantle of muskeg it was undoubtedly permafrost.

Overburden removal commenced in April 1954 with tractors and 8, 10, and 15-cu yd scrapers, some of which had been flown in for airstrip construction. The muskeg cover was removed quickly and easily. Once the underlying silt was bared, however, it thawed quickly, presenting a soupy, boggy surface with a frozen, icy base which offered difficult traction and poor digging. Progress was accordingly retarded.

With the opening of the shipping season and the arrival of more heavy equipment in late June, the tractors and scrapers were removed to the airstrip and replaced in the pit area by a 3-cu yd 101 Marion shovel converted to a dragline, and 14.8-cu yd Euclid trucks. It was necessary to construct temporary roads consisting of 3 to 4 ft of rock fill to support the equipment. Otherwise the operation was efficient, and in the five months between mid-July and mid-December, 185,000 cu yd of silt were removed. The bucket broke the frozen silt from the bottom and was thus able to handle all but the deeply frozen, massive portions. With the advent of winter and heavy frosts, the silt began freezing in the bucket and truck bodies. This was partly alleviated by heating the truck bodies with their engine exhaust, but by December 16 further operation was considered uneconomical and was suspended for the winter months. The dragline was converted to a shovel and used for rock removal in the open pit.



Plan of Gunnar open pit.

Except for a brief midwinter shutdown, rock work continued in the pit and benches were developed on the 700 and 670 elevations. When overburden removal was resumed in May 1955, it was decided to dispense with the dragline in favor of shovels only. Use of the shovel offered several advantages:

1) Rock fill roads on the silt, and the consequent double handling of the waste rock, were eliminated.

2) The shovel had more versatility than the dragline and could be used in silt or waste rock removal as the necessity arose.

3) Rock removal had in places advanced into the silt, so that convenient shovel sites with good floors were already available.

The shovels worked always from the existing bench floor. If the working face consisted entirely of silt, the operation was straightforward. When a rising floor presented a face of waste rock capped with silt, the two were removed together by drilling and blasting horizontal and low angle *lifter* holes into the rock face. Lifters were preferred to vertical holes, since the surface was very irregular and the silt had a tendency to cave and plug such holes. When the silt extended below the bench floor, it was necessary to reinforce the floor with rock fill, but this situation was not often encountered.

Contamination of ore with the glacial silt is undesirable due to the adverse effect of the latter in the mill circuit. Where silt overlay the ore, therefore, a more thorough removal technique was required. Where convenient, as much of the overburden as possible was bladed over bench faces to the shovels. The rest was washed with hydraulic monitors into natural sumps, and slurry was pumped into the lake.

Two 800-gpm Cameron centrifugal pumps supplied water through a 10-in. line to a 1200-gpm, P.R. & G. centrifugal booster pump, thence via a 10-in. line to the nozzle. Head pressure at the nozzle was about

220 psi, although pressure varied with the size of nozzle used, which in turn was governed by the rate at which the slurry was removed. Generally, one monitor with a 1¾-in. nozzle was used for cutting the overburden, and a second unit with a 1½-in. nozzle agitated the slurry. The latter was removed by two 800-gpm, 8x6 Vac-Seal rubber-lined centrifugal pumps through a 10-in. line. The slurry was discharged into a muskeg about 1000 ft from the lake and at only slightly higher elevation. It was hoped that much of the suspended solids would thus be strained out before the discharge reached the lake, but the settling rate was so slow that this was not very successful.

The slurry carried about 15 pct solids for a monitoring capacity of 40 cu yd per hr, and approximately 15,000 cu yd of silt were removed.

At the end of September 1955, 390,000 cu yd of overburden had been removed, distributed as follows: scrapers, 80,000 cu yd; dragline, 185,000 cu yd; shovels, 110,000 cu yd; and monitor, 15,000 cu yd.

The time involved in each operation is intentionally omitted since these figures are not meant for comparison of performances. Such comparison is not justified since different equipment served different purposes, and the functions of each are not always interchangeable.

An estimated 35,000 cu yd of silt remain to be moved on the 670 bench.

Drilling and Blasting: In addition to the overburden, more than 1 million tons of surface rock had to be removed to ready the mine for production. This included excavation of 150,000 tons for mill and mine building foundations, and an additional 900,000 tons to prepare benches in the pit before larger production machines could be profitably employed. The constant moving and rough terrain required a light, mobile machine but one capable of substantial production. Accordingly, six I-R X72 wagon-mounted air drills were acquired and equipped with tugger hoists. Air was supplied by I-R Gyro Flow 600-cfm mobile compressors. Drill steels of 1¼-in. diam and in 9, 18 and 27-ft lengths were made up on site and mounted with 2-in. tungsten carbide bits.

To reduce the original surface to the first bench level at elevation 700 ft from a maximum elevation of 745 ft required a two-stage effort on the part of the small drills. Since much of the original surface stood at or near 720 ft, and since the efficiency of the wagon drills was greatly reduced over depths of 25 ft, a subsidiary bench was established at 720 ft.

On bedrock surfaces, holes were drilled vertically with burden and spacing varying according to depth:

Depth, Ft	Burden, Ft	Spacing, Ft	Overdrill, Ft
Up to 8	4	4	2
8 to 14	4	5	2
Over 14	5	5 or 6	3½

In general, the 6-ft spacing and 5-ft burden was the maximum allowable, the criterion being a level floor rather than fragmentation. Fragmentation was not always good, however, since frost-widened fractures were abundant in the surface rock. Where excessive overburden or a rugged surface prohibited vertical holes, horizontal and low angle lifter holes were drilled, and where a face was of such height that vertical holes could not make grade, a combination of both vertical and lifter holes was employed. For the latter, the burden and spacing of the vertical holes were increased to 6 ft by 7 ft. Lifters were spaced at 5 ft with maximum burden of 7 ft. Toe holes were drilled below grade for a length of 8 ft, and the maximum length of the lifters was 27 ft.

The 2-in. blastholes were loaded with Forcite at the bottom, and Dygel or free-running ammonia dynamite above. Forcite was favored at the base of the hole because of its superior water resistance and its greater density, since the gage loss on long holes was up to ½ in. Use of Dygel or the ammonia dynamite in the top of the hole depended on whether the hole was wet or dry, and in the case of lifters, on the angle at which the hole was drilled. The amount of stemming in the top of each hole was governed not only by length of hole but by proximity of buildings and consequent need to control fly-rock.

Shots were fired with short period delay detonators, the priming cartridge being placed near the top of the column. In blasting to an open face the pattern was an increase of one delay period in each succeeding row from the face, with end holes one delay higher than the rest of the row.

Whenever feasible, shots consisted of six rows of holes drilled in a diamond pattern. Where a restricted working area necessitated drilling off a second shot before the first one was fired, it was found that missing one row of holes behind the first shot was sufficient to prevent loss of holes due to overbreak.

Production figures for the 2-in. machines during 1954 include primary and secondary drilling and blasting, overdrill, and stuck steels:

Footage drilled per machine per hr	21.8
Footage drilled per 2-in. bit	455
Footage drilled per ft of steel consumed	43
Tonnage broken per ft of hole	1.80
Tonnage broken per lb of powder	2.9

Since the bulk of this drilling was in waste rock, the figures give performance in the granite gneiss. Drilling in ore was 20 pct faster than in waste, giving a corresponding increase in bit and steel life.

With the pit surface reduced to elevation 700, and a floor and face established on the 670 bench, the overall pit plan was initiated. As discussed earlier, this plan calls for development in 30-ft benches, leaving a 21-ft berm on each bench. In addition to satisfying standing-wall requirements, the 30-ft wall provides a convenient working and scaling face for 3-cu yd shovels. Since the wagon drills are not efficient at such depths, they have been replaced by three I-R DM-RD Drillmaster blasthole drills with mounted 600-cfm diesel compressors.

These machines drill a 4-in. hole and under favorable conditions will perform at about 25 ft of hole per hr. Production figures for the three-month period beginning November 1955, under winter conditions, are:

Footage drilled per machine per hr	19.8
Footage drilled per bit	975
Footage drilled per ft of steel consumed	342

An 11-ft spacing and 9-ft burden, with 4½ ft of overdrill, is satisfactory for fragmentation and floor requirements. Crews are now experimenting with smaller burden and spacing to test if increased shovel and crusher production would be sufficient to compensate for the corresponding increase in costs.

The blasting technique has been varied slightly in the case of the 4-in. holes. Half the total charge is loaded and tamped hard into the lower one third of the hole. This consists of Forcite or Dygel, de-

pending on the wetness of the hole. The remainder of the hole is deckloaded, usually with free-running ammonia dynamite, to within about 3 ft from the surface. Drill cuttings are used for stemming.

Detonation is by reinforced Primacord which in turn is fired by a short period delay detonator for each hole. The delay pattern is the same as with the 2-in. holes but with two delay periods between rows.

Secondary drilling and blasting, when required for blockholes or high pit floors, is accomplished by pluggers and a low velocity powder, usually Stopite, the latter being used in an attempt to minimize flyrock. Sand blasting has also been employed effectively, but the size of the blast is restricted due to the concussive effects of such blasting on neighboring buildings.

Bench floors are frequently covered with up to 2 ft of drill cuttings and shot rock. During winter this is no problem since frost penetration keeps them rigid. In summer it is necessary to case the upper 2 ft of hole to prevent loose earth from plugging it.

All loading and transport is accomplished with two 3-cu yd 101 Marion shovels and seven 14.8-cu yd 36 TD Euclid trucks. Ore is hauled directly from the muck pile (or stockpile) to the primary crusher in the mill.

Winter Operations: Winter operations in an open pit at this latitude (59° 22′ N) may seem formidable. However, limited work carried out during the past winter, with the average day temperature at −10°F, suggests that no unusual difficulties will be met.

Except for a three-week shutdown over Christmas, drilling continued with restricted crews throughout the winter. Temporary lay-offs were necessitated by a combination of high winds and very low temperatures on only five days. Loading and hauling were suspended for lack of broken rock.

At the time, wagon drills were being used to level the original rock surface. To eliminate unnecessary and somewhat treacherous moving over snow-covered, uneven terrain, and to facilitate servicing, a housed and heated compressor station was set up, and air was fed to the drills through a 4-in. Victaulic pipeline.

Although such stations were necessary to protect the machines from extreme temperatures particularly when they were working only part time, they nevertheless had disadvantages. During freeze-up and break-up periods the air moisture content is very high, and at such times low temperatures produced icing conditions within the machines. This was eliminated by inserting an after-cooler into the airline, and adding a frost inhibitor. Despite the use of polar lubricants, a fire-bucket was frequently required to loosen the machines prior to drilling.

The 4-in. drills now in use move on crawler tracks and have the compressor mounted as an integral part of the machine. Movement, particularly on level floors, is accordingly facilitated, and the mounted compressor supplies hot air for the drill and warmth for the crews.

Lest unforeseen difficulties should produce a slowdown in pit operations during the present winter, ample ore has been stockpiled to ensure that the mill can continue to work at capacity. By the winter of 1956-1957, underground development will be at the point where ore can be withdrawn if required.

Sampling: Initial tonnage and grade calculations were based on results of vertical diamond drillholes at 75-ft centers. Needless to say, such data do not precisely define the outer limits of ore. As mentioned earlier, ore can seldom be recognized as such visually. Thus sampling is doubly important, not only to ascertain grade but to separate the ore from unmineralized syenite.

The following sampling procedure has proved both simple and reliable:

1) Blastholes are probed with a drillhole Geiger counter and results recorded at 5-ft intervals.

2) Cuttings from the blastholes are riffled down to make a 5-lb sample. This sample is checked with a Geiger counter equipped with rate meter.

3) If either step 1 or 2 indicates the possible presence of ore, the cuttings sample is assayed radiometrically.

4) After the rock has been broken, grab samples are taken as a rough check on previous results.

This procedure is begun well beyond the outside limits of suspected ore and continued through the orebody proper.

By this method, the frequent radiometric assays provide a constant check on the rate meter and drillhole Geiger counters. The experienced operator familiar with his equipment can thus decide immediately on completion of the blasthole probing whether the rock is to be broken for ore or waste.

Transportation: Generally speaking, mining operations in the Canadian North differ from their counterparts elsewhere only in the problems of severe winter climate and poor transportation. The winter problem, as it affects mining at Gunnar, has been discussed. The transportation problem has likewise been overcome.

Access to the area is by air or water only. Nearest railhead is at Waterways, Alberta, 225 miles to the southwest. From Waterways, Northern Transportation Co. Ltd. provides water freighting service into Lake Athabasca. Of necessity, the bulk of freight comes in by barge, and since the lake is open to shipping for only five months of the year, orders must be placed well in advance of expected delivery and must anticipate a year's needs. Oil requirements alone for heat and electric power amount to about 1½ million gal, or more than 7,000 tons of freight per year. At the close of the past shipping season, more than 50,000 tons of freight will have been delivered to the Gunnar dock since 1953.

Air freight similarly is a big item and is flown from Edmonton, Alberta, nearly 450 miles to the southwest. In November 1954, Gunnar completed construction of a 5600-ft airstrip. Prior to that aircraft landed on lake ice or water. About 1300 tons of freight have been flown in to date, of which about one third was landed on ice strips.

In spite of these obvious difficulties, production was achieved in little more than three years from the date of discovery. The property was staked in July 1952, and a 1250-ton capacity mill was completed in early August 1955. In September of that year, Gunnar delivered its first shipment of concentrates to become, after Eldorado's Port Radium and Beaverlodge operations, the third major uranium producer in Canada.

Acknowledgments

Thanks are due G. A. LaBine, president of Gunnar Mines Ltd., for permission to publish this article, and to J. N. Botsford, mine manager; N. A. Grant, assistant mine manager; G. G. Gilchrist, consulting engineer; A. W. Jolliffe, consulting geologist; and the mine staff at Gunnar for technical assistance and constructive criticism.

The Lavender Pit

Smooth operation results from careful management of

- **PIT PLANT**
- **DRILLING**
- **TRUCK OPERATION**
- **TRUCK-RAIL HAULAGE**
- **BLASTING**
- **DISPOSAL DUMP**

by Warren T. Smith

THE Phelps Dodge Corp. Lavender pit is located in the Bisbee mining district near the town of Lowell, Ariz. A quartz monzonite porphyry exists as an intrusive stock and was the source of mineralization for the limestone replacement orebodies being mined by underground methods. Leaching of the upper portion of the stock, and precipitation of the copper as chalcocite on the finely dispersed unoxidized pyrite in the monzonite porphyry, formed a low grade ore zone below a leached capping varying from 50 to 300 ft thick.

Churn drilling starting in 1916 and supplemental churn drilling begun early in 1950 indicated a mineable area about 3100 ft long by 2400 ft wide. In general drilling was on 100-ft centers. The pit was planned with 50-ft benches and wall slopes varying from 1 to 1 to a flatter slope of 1 1/3 to 1 dependent on the character of the ground. Materials within the pit limits, using a 0.6 pct minimum grade cutoff, were estimated at 41 million tons of 1.14 pct copper ore, 31 million tons of low grade sulfides suitable for leaching with an average grade of 0.42 pct, and 70 million tons of waste.

Initial stripping operations were started in April 1951. By March 1954, 42 million tons of advanced stripping had been removed and the pit was ready for ore production. The concentrator, rated at 12,000 tpd, was completed in June 1954, when operations commenced with test runs of ore continuing through July and August.

Before stripping operations were started the highest ridge was 5500 ft above sea level. Excluding removal of the ridge, sixteen 50-ft benches will eventually complete the pit to the final bottom elevation of 4500 ft. Bench widths were planned with a minimum of 140 ft from center line to center line, and at least 2000 linear ft of exposed ore on the benches was required before a daily tonnage could be assured for operation of the concentrator.

Primary Drilling: Primary drilling during the first two years of stripping was done by churn drills using 9-in. bits. With 50-ft benches, blastholes averaged 61 ft deep or bottomed 11 ft below shovel grade. Single row shots were spaced at 21-ft centers and about 36 ft back from the previous holes blasted. Over 500,000 ft of drilling was at an average rate of 77 ft per drill shift and 78 ft per bit sharpened.

Churn drills were later replaced by rotary drills. The first rotary was placed in operation November 1952, and churn drilling was progressively curtailed until fully displaced early in 1954 by three rotary rigs. These drills are electrically driven, weigh about 55,000 lb, and use 35-ft square hollow steel rods with 7⅞-in. tricone bits. Rotation speed varies from 20 to 75 rpm and bit pressure from 26,000 to 35,000 lb, depending on the character of the ground. Footage per drill shift averages 300 ft and bit life about 1620 ft. Sixty-foot holes bottom about 10 ft below grade. Spacing depends on the character of the ground and varies from 12 to 20 ft, averaging 15 ft. Three and one half shifts of rotary drilling are adequate for six shovel shifts or a daily mining rate of 37,500 tons.

Rotary drills are moved under their own power for short distances. For long moves on the same bench or those to another bench, a rugged hitch built into the front end of the rig permits towing with a tractor. Four guy wires from the top of the mast to the four corners of the drill frame eliminate the need for lowering the mast. Switch houses and power trail cables are maintained on all operating benches to eliminate loss of time when connecting up, following a long move.

Blastholes are sampled by a narrow metal cutter can with the open end placed adjacent to the hole

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being drilled. Normally two 25-ft samples are taken from a hole, each weighing 20 lb.

Dust control with the rotary drills is accomplished by introducing water through a needle valve into the compressed air line leading to the swivel at the top of the drill rods. The moisture drops the dust and the compressed air raises the drill cuttings to the collar of the hole. A 5-ft cone-shaped bonnet mounted under the drill frame deflects the cuttings downward to form a circular ridge around the hole. This material is used as stemming when the holes are blasted.

Explosives: Choice of explosives depends on the type of rock and whether the holes are wet or dry. Bag powder is used in dry ground of average hardness and 60 pct ammonia gelatin is used in hard ground and for wet holes.

Each hole is double primed with 25-lb, 6-in. cartridges of semigelatin. One primer is placed at the bottom of the hole and the other at the center of the charge. Wet holes in soft to moderately hard ground are loaded entirely with 25-lb semigelatin cartridges, which usually are lowered into the hole; however, if the powder rise is excessive, the cartridges are cut and the powder poured into the hole. Sixty pct ammonia gelatin is used in hard rock either wet or dry. Sticks are cut with a wooden paddle and the pieces dropped into the hole.

The loading ratio varies from ½ lb per yd of burden to more than 1 lb. The explosive charge per hole varies from 400 to 700 lb. Holes are double-primed with reinforced primacord with the ends either tied to a rock and lowered to the bottom of the hole or to a primer, if a primer cartridge is used. Single row shots of 30 to 40 holes are generally blasted at one time and break about 75,000 tons of rock. Millisecond delays are used only when blasting may affect nearby surface structures. The tons broken per pound of powder used have averaged about 4.5.

Secondary Blasting: The amount of rough muck depends greatly on the character of the ground and slips or shear zones. Ore delivered to the primary crushing plant must be reduced in size for a 48-in. gyratory. Stripping disposed of to dump sites requires less secondary blasting, and boulders weighing up to 10 tons may be loaded on the 25-ton capacity trucks. Secondary drilling is done with jackhammers, the pops loaded with 40 pct 1¼x8-in. ammonia stick powder and shot by primacord.

Loading: Shovel equipment includes two 5-yd and three 6-yd electrically operated units. Power is transmitted at 44,000 v to two transformer stations near the pit, each station serving separate pit circuits. Feeder lines distribute the power at 2300 v to portable switch stations located at strategic points on the various benches. A ground fault system is incorporated in the electric distribution system for drills as well as for shovels.

Power trail cables 800 ft long are neoprene-covered three-conductor with ground wires. When moving shovels, the trail cable is handled by wooden handled tongs. Cable crossings, at haulage roads, consist of two 4x12-in. 18-ft planks covered with ¾-in. plate 30 in. wide to allow a 6-in. gap underneath for the cable. A small load of fine material is dumped on the crossing and smoothed up to form a slight ramp over the cable. Crossings are moved with a bulldozer by passing a steel cable over the blade and hooking to each end of the plate decking. Whistler-type manganese steel teeth used on the shovel dippers have an average service of 185,000 tons. Discarded 1⅜-in. mine hoist cable used on the 5-yd shovels handles about 138,000 tons. New 1¾-in. cables used on the 6-yd shovels have an average life of 416,000 tons.

Yearly shovel loading rates during the five years of operation have varied from 5150 to 6130 tons per shovel shift for the 5-yd shovels and from 6180 to 6965 tons per shovel shift for the 6-yd machines.

The shovel crew consists of an operator who is responsible for the equipment and crew; an oiler who lubricates and keeps the shovel clean and attends to the trail cable and bank sprays; and a pitman who spots trucks, removes rocks from the roadways, and helps handle the trail cable in shovel moves. A rubber cone-shaped marker is used on each side of the shovel to assist the truck drivers in the proper spotting for loading and to keep the pitman at a distance from the face of the bank.

Truck Haulage: The coarse crushing plant is located in the north wall of the pit at an elevation of 5050 ft. Locus of ore is about the 4900 level. With comparatively short hauls and few adverse grades, 25-ton capacity single rear axle trucks were selected because of their flexibility in spotting at shovels and backing into the dumping position at the crushing plant and at waste disposal areas.

The fleet consists of 27 trucks of 25 tons each, equipped with 300-hp supercharged diesel engines. Twenty-one have single-stage torque converters and six are equipped with two-stage converters. The overdrive or fifth gear of the five-speed transmissions was found needless with hauls of less than 5000 ft and the fifth gears were therefore blocked off. The dual rear wheels are equipped with 18x24 or 18x25 tires and the front wheels use 14x24 tires. Roadways are kept in good condition by motor graders and wet down daily by 1600-gal capacity sprinkler trucks equipped with small gasoline driven pumps, which deliver a better spray than by gravity feed. Various makes of tires are used and tire mileage varies from 7920 to 12,060 miles per tire.

Trucks delivering ore to the crushing plant carry about 28 tons payload, and with an average haul of 2500 ft, a truck averages 1000 tons per 8-hr shift. Most of the stripping is delivered to a loading dock for transfer to a more distant dump by rail haulage. With this material, loads average 25 tons, and with a haul of 3450 ft, trucks handle about 1030 tons per shift. When rail haulage delays require delivery of waste to nearby disposal areas, trucks will average 925 tons a shift with a 30-ton load and a 5440-ft haul. The average lift approximates 130 ft.

Tubeless tires are being tried on a few trucks and apparently give satisfactory service with lesser costs for tubes and flaps. Engines in two of the trucks have been equipped with turbochargers but offer no apparent advantages at the altitude involved or with the small amount of adverse grades. Hydro retarders installed on two trucks have proved unnecessary with the minor amount of downgrade hauls and apparently increase truck maintenance.

Combination Truck-Rail Haulage: Available dump areas adjacent to the pit were insufficient to take care of all stripping and provide a suitable site for future leaching operations. The only available location to accommodate some 68 million tons of waste and low grade leaching material was a valley a mile and a half from the pit.

The problem was complicated by urban and industrial facilities and two heavily travelled highways between the pit and the proposed dump site which required overpasses. Rail haulage from the edge of the pit to the dump was chosen in preference to truck haulage based on length of haul and the amount of material to be disposed of daily during the major stripping program, when 10 shovel shifts necessitated disposal of 60,000 tpd.

The two highways were bridged with steel girder-type overpasses decked with 10x10-in. treated timber. The longer one is 472 ft, crossing three railroad tracks and a state highway. The shorter structure is 169 ft and carries the railroad over a county highway. They were designed for single track operation, erected in 1952, and the decking was overlaid by a 4-in. plank treadway for initial trucking to prepare the site.

The loading dock, necessary for transfer of muck from trucks to 43-cu yd side dump railroad cars, is an earth-filled structure 75 ft wide by 424 ft long. Width provides ample room for truck turning and backing up to the loading block, and length permits loading ten cars without spotting.

Tracks are arranged at both sides of the dock to permit loading a string of empties while the one locomotive in service delivers the loaded train from the opposite side of the dock to the dump and returns the empties while the other ten cars are being loaded. Under normal conditions 12 trucks serve two shovels in the delivery of material to the loading dock. The rail haulage has a capacity of 12,000 tons per 8-hr shift. The entire dock area is illuminated at night by flood lights mounted on 60-ft poles and the track by spot lights mounted in steel boxes attached to the dock side walls. A signal light system is used to divert trucks to nearby dump sites in the event of a delay or derailment of railroad cars.

Spillage at the loading dock is cleaned from the tracks by a rubber-tired dozer or front-end loader and placed back in the cars by the loader equipped with a 1½-yd bucket. A loading trap was constructed on one side of the dock to permit direct loading of spillage into a truck by means of a bulldozer. This method works very well, but the amount of spillage does not justify a second below-surface loading trap on the other side of the dock.

A 1200-hp diesel electric locomotive handles a ten-car train of approximately 1150 gross tons. Loaded trains are pushed from the loading dock to the dump area with the brakeman riding the lead car, which is equipped with a partially closed-in riding platform. When the train reaches the dumping area, the dumpman assumes control of all train movements and gives all signals while he and a helper do the actual dumping. While the train is being dumped, the brakeman walks back to the locomotive and rides the engine back to the loading dock. Here a cutoff is made, and the locomotive is run around to the other side of the dock to couple up to the loaded string of cars.

The main line and yard tracks are laid with 90-lb rail on treated gum ties spaced at 22-in. centers and slag ballasted. All turnouts are equipped with No. 7 cast steel frogs, 15-ft switch points, and manually operated switches, except for an automatic self-aligning stand at the junction of the two loading dock tracks. Rails on the overpasses were laid directly on the timber decking. The second or shorter overpass was later relaid with 133-lb rail because of heavy grades and curvature between the two overpasses and was slag ballasted instead of being spiked directly to the decking.

Disposal Dump: In preparing the dump for rail haulage a narrow fill 100 ft wide was made by truck haulage. This was sufficient for two dump tracks on each side—the northerly two for the disposal of waste and the southerly two for low grade leach material. As the dumps progressively widened, the two tracks near the easterly end were connected together to permit lengthening the dump and provide access to the two end dumps from either the north or the south trackage.

Dump tracks are 90-lb rail laid on untreated gum ties. In disposing of material from a newly moved track, a light plug is made the full length until the shoulder starts to build up. Then, starting at the far end, all cars are dumped in one place until the mound is high enough to interfere with the car doors. Spotting of cars is progressively moved until the mound extends the full length of the dump.

To level off the mound prior to track throwing, the track is broken into several sections and pulled back from the dump edge a distance of 10 to 12 ft by a crawler-mounted front end loader with two hooks bolted to the front of the 1½-yd bucket. These hooks are dropped over the ball of the rail and lifted until the ends of the ties are clear. The track is then pulled back about 2 ft at a time. The mound is leveled off by means of a bulldozer and the rough material pushed over the edge. Bulldozing is followed by finished grading, which is done by a motor grader, and the outside edge is sloped so that after the track is moved into place the outside or dump rail will be about 4 in. higher than the inner rail. The track is then pushed back to the edge of the dump for a new dumping position, the various sections are connected, ties are straightened and respiked if necessary, and the dump is again ready for use. No ballast is needed to prevent track displacement when cars are being dumped. This method of track throwing is much simpler than the method first used, when dump tracks were ballasted and thrown by track shifters. Although the front end loader is generally used in moving the track because it is smaller and not as rough on the track as a bulldozer, a dozer is used when the loader is in use elsewhere. Throwing of track and respiking rail by any method results in considerable tie replacements. At present, plate clamps on each side of the rail bolted down to a plate under the tie are being tried to increase tie usage and eliminate the need of gage rods.

Pit Plant: Surface facilities consist of an office building, change room, warehouse, lubrication shop, car repair shop, and general repair shop, all located near the easterly edge of the pit. Except for the office, all buildings are of steel frame construction covered with corrugated iron.

The floor of the repair shop, 80x220 ft, is served by a 25-ton overhead bridge crane for general repair work on trucks, bulldozers, locomotives, and other heavy equipment. A mezzanine floor 22 ft wide and the length of the building is used in rebuilding truck engines, torque converters, transmissions, and other lighter units. Machine shop equipment is on the ground floor under the mezzanine. This aisle is served by a 5-ton bridge crane.

All machine work and unit rebuilding is done on day shift. General maintenance is on both day and afternoon shifts. Field repairmen are assigned to all three operating shifts. The service truck used by field repairmen is equipped with two-way short wave radio so that it may be contacted quickly at any time.

The lubrication building services all pit equipment and also mobile equipment used by the underground mine mechanical department. Field servicing on each of the three shifts is performed by the use of a 10-ton truck equipped with tanks for diesel fuel, gasoline, greases, and oils used by the various pieces of equipment. A small gasoline powered compressor operates the pumps for greases and oils and supplies compressed air for displacing the fuels.

ABOVE: Truck loading operation in the Lavender pit, Bisbee, Ariz. A 6-cu yd shovel, 25-ton truck with 18-cu yd body. BELOW: Truck dumping at coarse crusher.

Major Pit Mining Equipment

Description	Size	Number
Electric shovels	5-yd	2
Electric shovels	6-yd	3
Haulage trucks	25-ton	27
Diesel electric locomotives	125-ton, 1200-hp	2
Side dump railroad cars	43-cu yd	24
Crawler mounted bulldozers		5
Rubber tired bulldozers		2
Motor graders	No. 12	2
Rotary drills	7⅞" Bit	3
Wagon drills	3½" Piston	5
Jackhammers	55-lb	20
Portable compressors	315 CFM	4
Rotary compressor—Skid mounted	210 CFM	1
Mobile unit—Mounted compressor	105 CFM	1
Water wagon trucks	10-ton, 1600-gal	2

Mounting and changing of tires is done in a small lean-to at the end of the repair shop. No sectioning or recapping is done in the tire shop. Tires needing repairs are sent out, and those for the 25-ton trucks are not recapped.

Shovel repairs are on a two-shift basis, and rotary drill repairs are performed on day shift only. These field repairs are under the supervision of a field foreman and, if possible, assemblies needing repairs are replaced and brought into the shop for correction. This unit repairing has been developed to a stage where a brake relining job on heavy trucks is done by removing the entire rear axle assembly and replacing with another reassembled unit. This method results in fewer delays to operating equipment and eliminates a tendency to do a fast but poor repair job.

Repair parts and tires are well stocked in the warehouse next to the repair shop. All items are card indexed and an automatic replacement plan is followed to prevent overstocking of other items infrequently used or delays caused by shortage of parts.

The White Pine Concentrator

by W. A. Hamilton and V. Lessels

WHITE Pine Copper Co. is a wholly owned subsidiary of the Copper Range Co. The property of White Pine is about 5 miles south of Lake Superior in Carp Lake Township, Ontonagon County, Mich. The new town of White Pine, adjacent to the company plant, is 18 miles south and west of the town of Ontonagon, and about 55 miles north and east of Ironwood, Mich.

A diamond drilling program begun in 1937 to develop the orebody was carried on until 1950. During this period a test shaft, known as the Schacht shaft, was sunk on the orebody and approximately 8000 ft of development drift were driven. In an effort to solve the ore beneficiation problems, considerable metallurgical test work was done on ore extracted through the mine development. This work was conducted in a small pilot plant located in the mill of the Copper Range Co. at Freda, Mich.

Active development of the White Pine mine had been considered in 1942. At that time the U. S. Government asked the Copper Range Co. to study the feasibility of at least partial development of the orebody at White Pine in view of the needs of copper for war. However, by the time suitable data were prepared and agreements were reached, the war took a turn for the better, and because it was believed that White Pine copper could not be made available in time to relieve the pressure of these war needs the project was abandoned.

At the outbreak of the Korean sitation, a revival of the project culminated in a loan of $57,180,000 by the Reconstruction Finance Corp. The decision was made to proceed on March 1, 1952, with detailed planning of the project. Design and engineering organizations were grouped in Chicago, and simultaneously field organizations were set up at White Pine, Mich., to commence on the problem of actual construction. The Turner Construction Co. had been retained by the Copper Range Co. for the preliminary studies made in 1942 and was retained as agent for the White Pine Copper Co. at the later date. Under this arrangement the forces of the Western-Knapp Engineering Co. of San Francisco were retained for the mine, mill, and smelter design; Stone and Webster of Boston was asked to design the power plant; Pace Associates of Chicago was given the task of planning and design of the townsite and related facilities as well as nonprocess buildings, and the firm of Moran-Proctor-Muesser & Rutledge of New York City was retained as consultant on soils and foundations design.

Meanwhile the ore beneficiation testing facilities at Freda, Mich., were revamped and an expanded program of ore testing and gathering of engineering data for plant design was instituted.

Mill design and pilot plant work proceeded at a rapid rate. By the end of 1952, only nine months after actual design had been started, the mill flowsheet was developed in final form and almost all the principal mill equipment was purchased, or specification for purchase had been prepared. Active construction of the mill started during the winter of 1952-1953. Construction continued during 1953 and 1954 and on October 5, 1954, the first section of the concentrator was placed on stream and the first phase of winning copper metal from the ore began. By late December all sections of the concentrator had been constructed, tested, and placed in operation. Since the early part of January 1955, the mill has been on a seven-day week, around the clock operation with all or part of the equipment in continuous operation.

The White Pine orebody lies near the bottom of the Nonesuch formation, which consists of a 500-ft layer of siltstones, sandstones, and shales. The ore-

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body is essentially four sedimentary beds, which have been labeled in their respective order as Lower sandstone, Parting shale, Upper sandstone, and Upper shale. Principal mineralization occurs in the shales. Mineralization consists mostly of chalcocite with possibly as much as 10 pct native copper. Chalcopyrite, bornite, covellite, and pyrite are present only as traces. The ore also contains silver, present chiefly as native silver. The silver-copper relationship is similar to the so-called *half-breeds* of the Calumet-Houghton area where both silver and copper are found unalloyed in their natural state. The ore minerals are extremely disseminated throughout the shale, and locking with gangue is present in all sizes visible under the microscope.

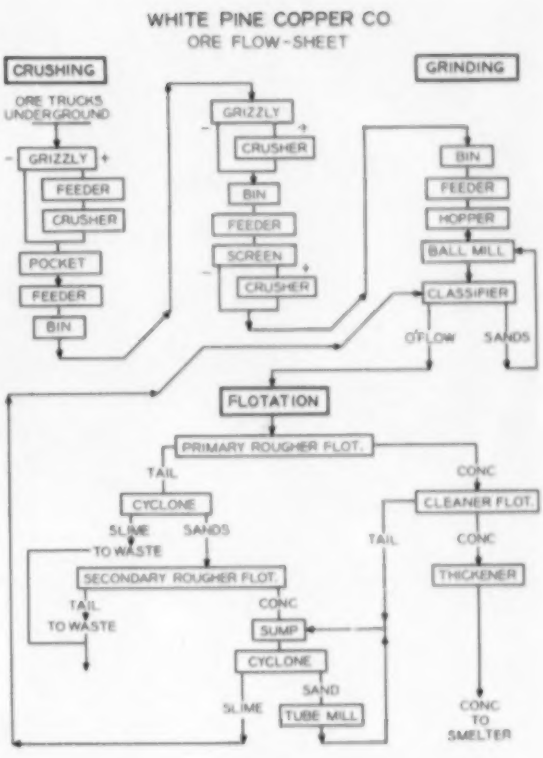
At the present time only the Parting shale portion of the orebody is being mined. Research is in plan to develop methods to extract the Upper shale orebody. The mining method is a modified room and pillar system. Drilling by jumbos is standard practice, followed by blasting and loading with Joy loaders to diesel-powered trucks for transportation to the underground ore pocket.

The Process

Crushing Practice: The first stage of crushing is accomplished underground. The run-of-mine ore is delivered by diesel truck to grizzlies set at 10-in. opening. The oversize is fed to two 32x40-in. jaw crushers operating in parallel, set at about 6-in. jaw opening. The crushed oversize joins the grizzly undersize in a 3000-ton, live load capacity, underground storage pocket. At about 110 ft below the haulage level, ore is fed by chute and apron feeder to a speed-up or conveyor feed belt. The ore, which is now essentially −8-in. maximum size, passes on to a conveyor system to the surface crushing plant. The mine conveying system consists of 54-in. wide upper and lower flight conveyors, operating in series, each driven by three 250-hp motors. Two of the electric motors operate in tandem on the same drive pulley and the third motor operates a separate drive pulley. This belt system has a capacity of 2400 tph. The lower flight is 1650 ft between centers and the upper flight 1750 ft between centers. A belt scale is located on the lower flight conveyor to check mine tonnages delivered to the surface. At the surface, two 54-in. conveyors in series carry the ore from the end of the mine conveying system to the top of the surface crushing plant. At this point the ore is diverted by a short shuttle conveyor into either of two coarse ore storage bins. All the surface crushing plant bins are reinforced concrete silo-type structures, 105 ft high and 32 ft ID, with a live load capacity of about 1500 tons each. There are two coarse ore bins and three intermediate size ore bins all in line. These bins were poured in one-piece type concrete construction.

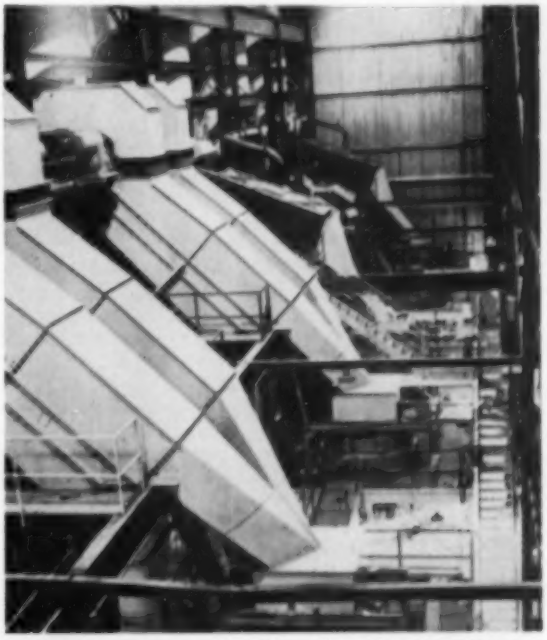
Coarse ore is fed from the bottom of the two coarse ore bins by four 3½x16-ft apron feeders to stationary grizzlies set at 2½-in. maximum opening. The oversize falls by gravity from the grizzlies through chutes to two 7-ft Symons standard crushers operating in parallel, set at 1¼-in. opening.

The combined crusher product and grizzly undersize is conveyed by a 54-in. wide scissors belt arrangement to the top of the crushing plant, where it is distributed over three concrete intermediate size ore bins by a 54-in. tripper conveyor arrangement. Intermediate size ore is fed from the bottoms

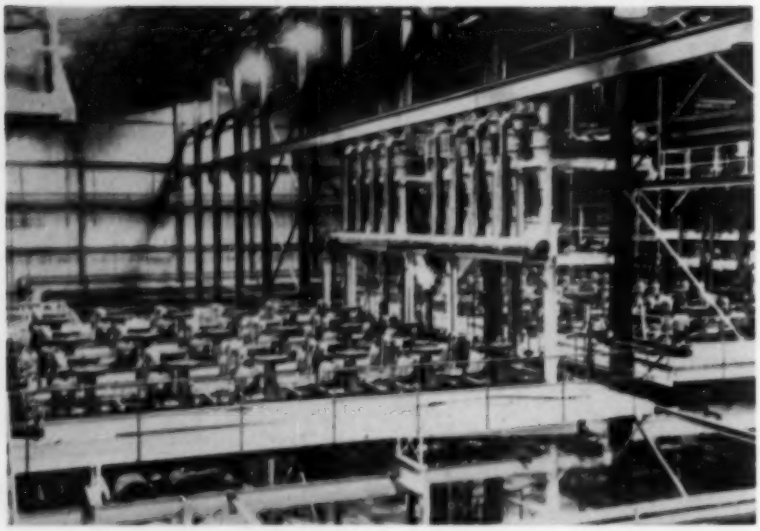


of these bins by 4x14-ft vibrating feeders. Each bin is equipped with one feeder which discharges to a 72x168-in. double-deck screen with 1¼-in. and ⅞-in. square top deck openings and ½x4-in. and ⅜x3-in. slotted opening bottom decks.

The screen oversize is fed directly to each of three 7-ft Symons shorthead crushers operating in parallel, set at 5/16 in. The screen undersize, which bypasses the crushers, and the fraction from the crushers



General arrangement of crushing plant. The two Symons standard crushers are in the foreground.



Desliming cyclones and secondary rougher flotation circuit in mill section No. 3.

then join and are conveyed to the fine ore bins in the concentrator section by a series of 54 and 60-in. wide belt conveyors.

A shuttle conveyor system, consisting of two 48-in. belts traveling in opposite directions, distributes the crushed ore to the fine ore bins. An automatic sampler and sample splitter cuts the ore stream prior to entering the fine ore bins. A belt scale records the weight of the ore delivered to the concentrator.

The entire conveying system from the mine lower flight through the crushing plant to the fine ore bins, as well as all the feeders, screens, crushers, and related equipment, are controlled by the crusher operator by means of push-button controls, arranged in flowsheet form on the panel. Telltale lights indicate equipment operation and position of shuttle and tripper conveyors with regard to bins. Ammeters for conveyor and crusher motors and speed indicators for belt feeders are located directly over the console panel for observation and corrective operational control if required. A telephone and call system, which extends from the underground crushing station to the top of the fine ore bins, covers all transfer points and principal locations along the conveying, crushing, and screening system.

The crushing plant operates two shifts per day seven days per week. A capacity of 1500 tph of finished material has frequently been attained.

At present the crushing plant operates in open circuit. An intensive study of crushing efficiency and an engineering study of a possible separate screening plant to operate in closed circuit with the shorthead crushers is under way.

Concentration: Crushed ore from the surface crushing plant is distributed by shuttle conveyors into six reinforced concrete silos, each 50 ft ID and 70 ft high. Under each concrete bin or silo, four vibrating feeders 1½x10 ft feed the fine ore to a 24-in. conveyor, which in turn delivers to the drum of the combination drum and scoop feeder at the feed end of each ball mill. The concentrator is divided into three sections with space sufficient within the present building for a fourth section if it is required. Principal equipment in each section consists of two ball mills, four classifiers, one tube mill, eighty-four 66-in. Fagergren flotation machines, eighteen 24-in. cyclone classifiers, six 12-in. rubber-lined pumps and reagent and pulp distributors, sump pumps, and related equipment. In each section six cyclones are normally in use in the desliming operation and five cyclones in the secondary grinding section. Only three main pumps are operated. Separate spare sumps and pumps with quick change sluice gates provide spare equipment in case of pump failure or for maintenance inspection.

The crushed ore discharges from the 24-in. con-

Crushing Plant Performance

	Feed to Shorthead Bins		Shorthead Screen Undersize		Feed to Shorthead Crusher		Shorthead Discharge		Feed to Fine Ore Bins	
Pct of Total	100.00		34.27		65.73		65.73		100.00	
Mesh	Wt, Pct	Cumulative Wt Pct	Wt, Pct	Cumulative Wt, Pct	Wt, Pct	Cumulative Wt, Pct	Wt, Pct	Cumulative Wt, Pct	Wt, Pct	Cumulative Wt, Pct
+2"	8.87				13.49					
+1½"	13.88	22.75			21.12	34.61				
+.742"	30.59	53.34	0.94		46.05	80.66	8.25		5.74	
+.525	9.00	63.34	7.00	7.94	11.41	92.07	18.53	26.78	14.58	20.32
+.371	8.92	72.16	18.93	26.87	3.70	95.77	24.13	50.91	22.35	42.67
−.371	27.84		73.13		4.23		49.09		57.33	

Feed to shorthead bins, shorthead screen undersize and feed to fine ore bins are averages of samples taken throughout September and October 1955.
Feed to shorthead crusher is calculated on the basis of 90 pct screening efficiency.
Shorthead discharge is calculated from the shorthead screen undersize and feed to fine ore bins.

veyor and enters the drum of the combination drum and scoop feeders of the primary grinding mills. Each of the six primary grinding mills is a Marcy grate type, 12 ft diam by 13 ft long inside the liners. These mills, to the authors' knowledge, are the largest primary grinding mills ever operated. Each weighs approximately 300 tons fully loaded with 100 tons of balls. A 150-rpm, 4160-v, 1500-hp synchronous motor furnishes power to the mill through two flexible couplings and a floating shaft connected to the pinion. The initial set of shell liners were Lorain rolled plate liners and Chrome-Moly lifter bars. Feed end liners are cast Ni-hard, and the grates are Chrome-Moly ⅝ to ¾-in. slots. Mill speed is 16.05 rpm, approximately 72 pct of critical speed. The double scoop has a radius of 90 in. Trunnion diam is 60 in. The bull gear is a single helical gear, 26-in. face, 215 tooth with 1-in. D.P. made in two halves.

The enormous weight of the loaded mills, coupled with structural problems, excluded the possibility of complete removal of mills for relining or other maintenance work. As a consequence, concrete jacking piers for trunnion bearing removal were incorporated in the foundation design. Other design features were adopted to insure ease of relining and other maintenance work on the mills in place.

Each mill operates in closed circuit with two 84-in. spiral classifiers. A splitter connected to the discharge end equalizes the load to each classifier. Circulating load in the classifiers varies with the feed condition but is normally 250 to 300 pct.

Although two primary grinding mills are normally operated per section, either mill may be down for inspection or repair and the remaining mill may operate without disadvantage.

Overflow from all four classifiers in each section is gathered and pumped to an eight-way distributor, which apportions the pulp to the primary rougher flotation. There are eight rows of 66-in. Fagergren flotation machines with four cells per row in each mill section. Froth from this primary rougher operation flows by gravity to the cleaning step, which is done in eight rows of 66-in. Fagergren flotation machines with two cells per row in each section. Froth from the cleaners is pumped to a central location, sampled, and transferred by gravity to the concentrate thickener outside the mill building. Cleaner tailings or middling product flows by gravity to the regrind pump sump.

Primary rougher flotation tailing flows by gravity to the desliming pump sump from whence it is pumped to a battery of 24-in. cyclones. The cyclone overflow or slime product is discarded as a tailing, and the cyclone underflow is refloated in six 6-cell rows of 66-in. Fagergren flotation machines per section. The resulting secondary rougher flotation tailing or sand tailing is discarded to waste. Rougher froth flows by gravity to the regrind pump sump, where it joins the cleaner tailing fraction. The combined secondary rougher concentrate and cleaner tailing is pumped to a battery of 24-in. cyclones in the regrind circuit. The regrind cyclone underflow or coarse fraction flows by gravity to a Traylor 8x22-ft tube mill. Tube mill discharge flows by gravity back to the regrind pump sump. The cyclone overflow is distributed to the four spiral classifiers in the primary grinding section. Because the regrind cyclone overflow is normally 6 pct solids, advantage is taken of this excess dilution by adding this fraction at the discharge box of the primary ball mills, where it forms a portion of the dilution water required in the primary classification system.

Primary grinding mills and classifier

Thickening and Filtering: The cleaned concentrate is gathered and pumped to a central location in the mill building, where it is sampled, and then flows by gravity to the center well of a 100-ft Dorr torque arm thickener with a concrete tank. This equipment operates in the open with a small shelter house over the drive mechanism. Thickener overflow is returned by pumps to the mill water system. Thickener underflow is regulated by spigots and flows by gravity to a pump sump, from which it is elevated to a splitter box in the filter building. From the splitter box the thickened concentrate may be diverted to either or both of two string discharge filters 10x½-ft diam x 18-ft face. Filtrate water is returned to the thickener tank, and the filter cake is discharged to a belt conveyor. The filter cake is weighed and sampled prior to discharge from this gathering belt. The filter cake, which contains a large proportion of gangue-sulfide grains, contains 18 pct moisture. The belt scale weighing the filter cake also regulates feeders for lime and pyrite in the smelter fluxing system so as to regulate the amount of flux proportional to the tonnage of filter cake production. The filter conveyor discharges to a cross belt carrying the smelter flux material as noted above. The combined filter cake and smelter flux is discharged from this cross belt into a pug mill, where thorough mixing takes place. Discharge from the pug mill is conveyed to a point in the smelter feed and bedding building. From this point the mixed charge may be diverted to storage or directly to the reverberatory furnace. Pyrite purchased from an outside source is required in the

smelter flux because White Pine ore is deficient in sulfur and iron content for matte formation.

Concentrator Performance

Design capacity of the concentrator is 10,500 dry tons of ore per day. Actual sustained capacity exceeds 11,500 dry tons per day, and with all grinding units operating under favorable conditions, more than 12,500 dry tons per day have frequently been achieved. Individual mill performance has exceeded 110 tph under favorable ore conditions.

Mill Metallurgy

Item	Wt, Pct	Copper, Pct	Distribution
Feed	100.00	1.198	100.00
Concentrate	3.25	30.28	82.17
Slime Tails	39.38	0.181	5.95
Sand Tails	57.37	0.248	11.88
Total Tails	96.75	0.221	17.83

Screen Analyses: Primary Classifier Overflow (Flotation Feed) and Filtered Concentrate

	Classifier Overflow		Filtered Concentrate	
Mesh	Wt, Pct	Cumulative Wt, Pct	Wt, Pct	Cumulative Wt, Pct
+ 65	0.38			
+ 100	2.00	2.38	0.28	
+ 150	5.45	7.83	0.93	1.21
+ 200	6.37	14.20	1.64	2.85
+ 325	10.66	24.86	4.93	7.78
− 325	75.14		92.22	

Performance of Desliming Cyclones

	Feed	Overflow		Underflow	
Solids, Pct	30	11		40	
Wt, Pct	100	38		62	
Mesh	Wt, Pct	Cumulative Wt, Pct	Wt, Pct	Wt, Pct	Cumulative Wt, Pct
+ 65	0.40			0.65	
+ 100	2.65	3.05		4.27	4.92
+ 150	7.31	10.36		11.79	16.71
+ 200	6.66	17.02		10.74	27.45
+ 325	10.00	27.02	0.70	15.70	43.15
− 325	72.98		99.30	56.85	

Pressure: 25 to 28 psi — Feed inlet: 4½ in. ID. Apex discharge: 3 in. ID. — Vortex finder: 4⅝ in. ID.

Performance of Regrind Circuit Cyclones

	Feed	Overflow		Underflow	
Solids, Pct	15.5	7.4		57.2	
Wt, Pct	100.0	52.1		47.9	
Mesh	Wt, Pct	Cumulative Wt, Pct	Wt, Pct	Wt, Pct	Cumulative Wt, Pct
+ 65	0.1			0.4	
+ 100	0.5	0.6		1.6	2.0
+ 150	2.3	2.8		5.2	7.2
+ 200	3.4	6.2		7.2	14.4
+ 325	10.3	16.5	0.6	24.8	39.2
+ 20μ	47.4	63.9	36.4	53.9	93.1
− 20μ	36.1		63.0	6.9	

Pressure: 36 psi — Feed inlet: 3⅛ in. ID. Apex discharge: 2½ in. ID. — Vortex finder: 4⅝ in. ID.

Reagents

The reagents in use are fuel oil, lime, Minerec *B*, sodium isopropyl xanthate, pine oil, Dowfroth, and American Cyanamid reagent 249. A small amount of Separan is used to assist flocculation in the concentrate thickener.

Dry and liquid reagents in drums are received by rail and stored in a convenient location in the reagent area in the mill. One man on duty one shift per day mixes and prepares all reagents and cleans and inspects reagent feeders and distributors.

Pine oil and fuel oil are received in tank cars and are unloaded into 25,000-gal underground storage tanks. Lime is received as dead burned pebble lime in covered hopper cars and is discharged into track hoppers and elevated into steel storage bins. Covered screw feeders and conveyors feed and convey the pebble lime to an 8x10-ft ball mill, where it is slaked with water and converted into milk of lime. The milk of lime is stored in four 25,000-gal storage tanks equipped with agitators.

All reagent feeding, with the exception of lime, is accomplished by Flowrators. The various reagents are circulated throughout the concentrator by means of the reagent piping system fed by fractional horsepower gear pumps. Circulating lines are equipped with breather pipes, so that they function also as constant head tanks ahead of the Flowrators. Where the reagent flow is to be distributed among a number of rows, the regulated stream of reagent is divided by distributors of Denver-Lewers type. The system has been relatively trouble-free and is neat and clean in operation and appearance in contrast to the usual unsightly array of reagent-encrusted tanks and open type feeders.

Reagents Used and Approximate Consumption in Lbs Per Ton of Original Feed

Reagent	Point of Addition	Consumption, Lb Per Ton
Lime	Concentrate thickener and tailing sumps	4 to 5
Sodium isopropyl xanthate	Stage-fed into flotation	0.20
Pine oil	Stage-fed into flotation	0.10
Fuel oil	Added to primary grind	0.50
Reagent 249	Head of secondary flotation	0.015
Minerec *B*	Added to primary grind	0.12
Separan	Concentrate thickener	0.0001
Dowfroth	Head of primary flotation	0.025

Minerec and fuel oil are fed to the primary grinding circuit. Dowfroth is fed to the classifier overflow sump. Xanthate is stage-fed into the circuit at the classifier overflow sump, the third cell of the primary rougher flotation, and the first, second, and third cells of the secondary flotation circuit. Pine oil is also stage-fed into the circuit in the third cell of the primary rougher flotation and the first and third cells of the secondary rougher flotation. Reagent 249 is added at the head of the secondary rougher circuit. Milk of lime is added to the concentrate pump sump and also to the tailings as they leave the mill to induce flocculation in these materials. Provision is made for lime addition to the ball mills, but normally the tailing return water use is sufficient for desired alkalinity. A small amount of Separan is added to the concentrate thickener along with the lime.

Grinding Balls

Grinding balls of carbon manganese forged steel are purchased in 3-in., 2-in., and 1-in. sizes. At

Ball Sizes and Consumption in Lb Per Ton of Original Feed

Ball Size, In.	Consumption Lb Per Ton Original Feed
3	0.765
2	0.094
Total Primary Grinding, In.	0.859
1	0.309
Total Grinding	1.168

present, the daily ball ration in the primary mills consists of 90 pct 3-in. balls and 10 pct 2-in. balls. In the secondary grinding circuit only 1-in. balls are used.

Grinding balls are weighed in buckets. The ball bucket is transferred by overhead crane to stands provided at the feed end of each mill. A simple hand lever and launder arrangement is provided at this point to discharge the balls into the drum feeder of the mill.

Tailings Disposal

The sand and slime fractions of the mill tailings flow by gravity in separate steel launders from the concentrator building to the tailing pump house, which is located about 350 ft from the concentrator. During the latter part of the flow, the tailing launders pass through a concrete-lined tunnel. In the tailing pump house, provision is made for the slime and sand tailings to be combined and diverted to either of two concrete sumps or to retain their identity and go to individual sumps. Two batteries of rubber-lined pumps, with four pumps to each battery, are located in this building. One battery of pumps is equipped with 200-hp motors and the other four pumps are powered by 125-hp motors. All pumps are furnished with variable speed *V* belt drives.

Tailings are pumped to a common header on each battery of pumps which joins 24-in. discharge pipes connected to each header. Ends of the discharge pipes extend 2000 ft and 5000 ft respectively east of the tailings pump house.

At White Pine the short summer season available for building dam and the lack of coarse material in the tailings seemed to exclude the normal method of tailings disposal, whereby the sandy fraction of the ore is discharged around the perimeter of the dam and becomes the material of which the dam is constructed. Another factor in selection of a method of tailings disposal was the flat terrain. Ground slope in the area is about 40 ft to the mile in the direction of Lake Superior, and tailing impoundment is required on a minimum of three sides.

Accordingly an earthen dam of local clay, 20 ft high and approximately 7700 ft long, was constructed about 1½ miles north of the mill. Wing dams 2000 and 3000 ft long extending at right angles to the main dam were also constructed of clay. Provision was made in the original design of the dam for an eventual height of 40 ft. One 10-ft lift over the 20 ft has already been accomplished. Four concrete decant towers equipped with wood weirs are located at convenient intervals close to the main dam. These decants are serviced by bridges from the main dam and also by outboard motor boat. The decanted overflow is conducted through the dam in steel culverts to a ditch that parallels the downstream face of the dam. This ditch conducts the overflow water to a basin where the tailing water return pumps are located. Tailing water is returned to the 250,000-gal constant head tank at the concentrator through 8175 ft of reinforced, prestressed, concrete pipe of 30 in. diam. A total of 15,000 gpm of pump capacity is installed in the tailing return water pump house. At a convenient point in the catch basin for the tailing water, a concrete spillway has been constructed. This takes care of surplus water due to heavy rainfall or melting snow during the spring run-off season. The spillway and the tailing dam have functioned well. During the last year a 4-in. rainfall in 8 hr was handled through the system without any difficulty.

Fresh Water Supply

The fresh water supply for the entire project, including the mine, mill, smelter, and townsite, is derived from Lake Superior. Near Silver City, Mich., approximately 5 miles north of the plant site on the banks of the Iron River, a 7 x 12-ft shaft, 110 ft deep, was sunk. A 5 x 7-ft tunnel was driven from the bottom of this shaft a distance of 2600 ft, traveling under the Iron River and out under Lake Superior. At the end of the tunnel a raise was driven to the lake bottom. The depth of the water at the lake opening is approximately 26 ft. This method of going out from the shore of the lake a considerable distance was necessitated by the heavy lake ice in the winter time. Water flows through a grid at the lake bottom entrance into the tunnel and shaft to concrete sump. Three Ingersoll-Rand type-10HLV pumps, 6000 gpm each at 315-ft total head and powered with 600-hp synchronous motors, deliver the water to the plant site through 24,500 ft of 36-in. reinforced, prestressed concrete pipe. Upon arrival at the plant site, a portion of the water is diverted to the water treatment plant for potable water needs of the plant and townsite. The remainder coming from Lake Superior goes into the power plant spray pond. Cold water from the spray pond is in continuous circulation through the steam turbine condensers in the power plant. The water returning from the condensers overflows a sump on its passage back to the spray pond. From this sump, three 6200-gpm pumps are available to deliver water to the 250,000-gal constant head tank located near the northeast corner of the mill building. Because condenser return water is usually at least 10° warmer than the lake water, some operational benefits are derived in the mill from the use of the warmer water. A portion of the lake water is diverted prior to entry into the mill tank for use in the concentrate launder spray system and also for use as gland seal water on the mill circulating pumps.

As demand varies, Telemetering system automatically regulates the lake pump station as well as the mill water pumps.

Total actual water use is slightly less than 1 gpm per ton milled per day. About 50 pct of the mill water requirements is supplied as fresh water. The remainder is returned from the tailings disposal area and from the area draining into the tailings basin. A large amount of contamination of tailing return water by decaying vegetation occurs, particularly in the spring and other wet seasons of abnormal precipitation and run-off. At these times volume of tailing return water must be carefully watched to prevent adverse metallurgical results in the concentrator.

Power Source, Distribution, and Usage

Power for the entire plant is generated locally in a plant of modern design. Three 15,000-kw Westinghouse steam turbines generate power at 13,800 v. The power plant is of unitized design with two turbine generators receiving steam from pulverized coal-fired boilers. Steam is delivered at 875 psi and 905°F. The third turbine is operated by steam from the waste heat boiler installation in the smelter. Power is distributed throughout the operation at 13,800 v and reduced at the point of usage. All power distribution is by means of conduits carried in underground tunnels connecting various buildings in the plant site. Steam for heating purposes is also carried from a separate boiler in the power plant through the same tunnels to the various buildings.

Power enters the mill buildings at 13,800 v and is reduced in transformer banks located within the buildings. All motors 200 hp or over operate on 4160 v. Motors under 200 hp operate on 480 v. Lighting and a very minor amount of motor equipment operates on 110 v. In the concentrator building an entire bay has been devoted to transformers, M.G. sets, and starting equipment. In the crushing plant the transformers and starting equipment are located under the concrete ore bins.

Power use in the milling operation averages almost 23 kw-hr per ton of ore milled. Approximately 1.71 kw-hr per ton of ore are consumed in crushing, conveying, and screening, exclusive of the mine conveyors. Approximately 21.26 kw-hr per ton of ore are consumed in grinding, concentrating, pumping, and water supply, thickening and filtration and lighting. The tailings pump house and the tailing water return pump house are electrically heated.

Mill Organization

The mill organization totals 113 people, including technical employees, supervisory employees, maintenance, laborers, and extra employees, for swing-shift operation on a 7-day week, 24-hr day basis. Salaried employees and staff total 24 and hourly employees total 89.

The mill dept. is divided into subdepartments: operations, technical, and maintenance.

Operational dept. supervision consists of a production foreman, four regular mill shift foremen, a spare foreman, and three crushing plant foremen. A total of 50 hourly employed men on all shifts comprise the balance of the operating crew.

The maintenance dept. consists of a maintenance supervisor, a concentrator repair foreman, a crusher repair foreman, and a labor foreman. A total of 37 hourly employed men comprise the crew, of which 23 are classified as maintenance men and 14 as common laborers. A salaried tool room clerk completes the roster of the maintenance section.

The technical dept. consists of a mill metallurgist, assistant mill metallurgist, two laboratory assistants, and four sample men.

All three subdepartments are directly responsible to the mill superintendent. A salaried mill clerk channels all information, keeps operational records and files up to date, and takes care of payroll sheets for the mill dept. All work in the mill dept. is normally scheduled on a 42-hr week basis.

Structural Details

All the buildings in the mill dept. are steel frame on reinforced concrete foundations with 20 gage, corrugated, iron panel siding with fiberglass insulation and built-up roofs. All buildings, except the tailings pump house and the tailings water return pump house, are heated by fin-type blower units. Steam for these heaters is furnished by a central heating boiler located in the power plant.

The crushing plant, including repair bay, is 24 ft 7 in. long and 69 ft 8½ in. wide. Maximum height is 123 ft 6 in. and total ground area is 14,180 sq ft. Three type-N Rotoclone dust collectors service the crushing plant and conveying system. A 40-ton overhead crane services the crushers and repair bay. All conveyor galleries are of steel frame construction with corrugated siding. Conveyor galleries are insulated but unheated. An elevator provides transportation of men between various levels of the crushing plant.

The mill building or concentrator is 458 ft long and 294 ft wide. The maximum height of 100 ft to the roof near the bin side slopes to about 38 ft to the roof on the west side of the building. Approximate ground area of the concentrator is 134,652 sq ft and approximate building space is 9,290,988 cu ft. The building is divided into seven bays running the full length. The bays are consecutively named, starting at the fine ore bins as follows: primary grinding bay, secondary grinding bay, cleaner flotation bay, primary rougher flotation bay, secondary rougher flotation bay, power bay, and a bay containing offices, change room, laboratory, and reagent area. All the transformers and M.G. sets and electrical starting equipment are located on two floors in the power bay. The bay containing the mill office, testing laboratory, change rooms, and sample preparation facilities also has sufficient room on the north end of the building to provide for reagent mixing and handling equipment. Reagent storage is provided on the floor directly above the mixing tanks in this section. Reagents are elevated to this upper floor for storage by means of a hydraulic lift-type elevator. A railroad track runs along the west side of the concentrator building, and from this location reagents and other material can be easily handled.

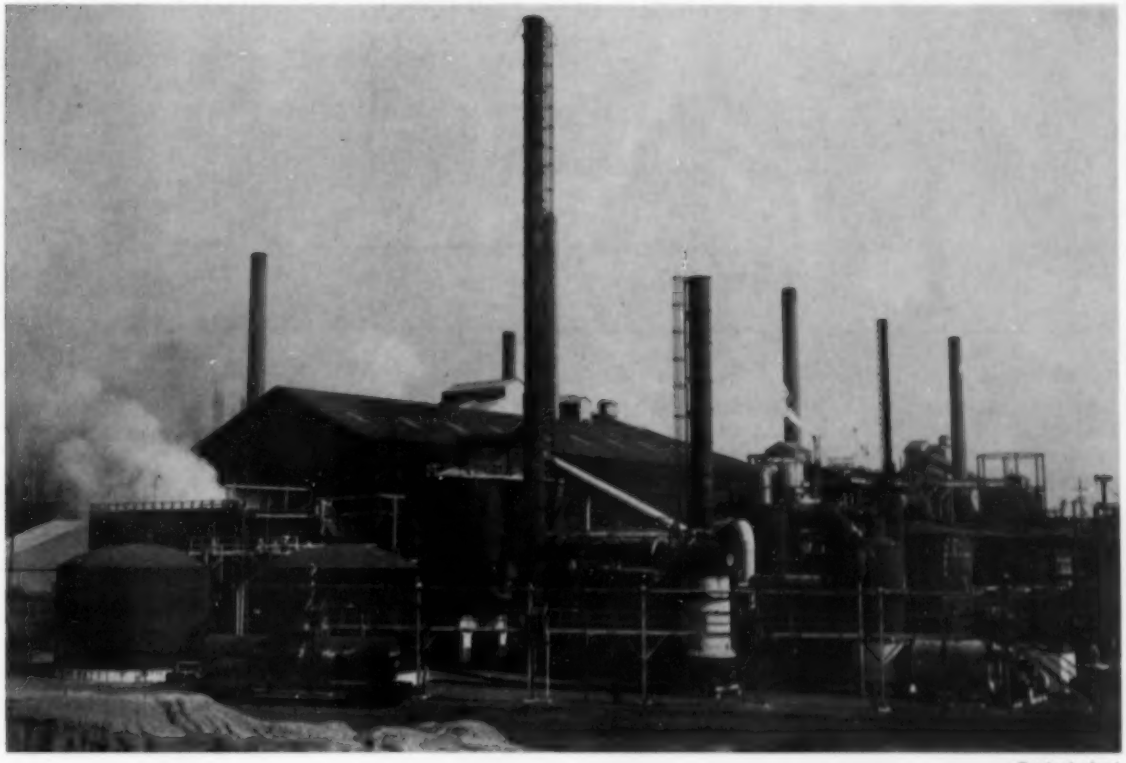
The concentrator repair bay extends the entire width of the building at right angles to the longitudinal bays and has a railroad track extending the entire length. This repair bay is on the south end of the building. On the north end enough room is available to put in an entire mill section, including ball mills, tube mill, and other equipment should it be required. At the present time this area is being used for storage and warehousing.

The primary grinding and secondary grinding bays are serviced respectively by 100-ton and 50-ton overhead cranes equipped with auxiliary hooks of 15-ton capacity. All the flotation bays are serviced by 10-ton overhead cranes.

The building is without windows except for offices and laboratory. Lighting is by means of mercury arc and incandescent lamps. Uncorrected color mercury lamps and incandescent lamps are arranged in clusters to approximate daylight conditions.

Acknowledgments

The writers express appreciation to J. P. Lally, president of the Copper Range Co. and White Pine Copper Co., for permission to publish this article and to the mill metallurgical and engineering staffs of White Pine Copper Co. for their aid in collecting and preparing data.



Contact plant

The Fluidized Bed Reactor Installation At Sparrows Point

by H. B. Scharf and E. C. Dominguez

THE Sparrows Point plant of Bethlehem Steel Co., southeast of Baltimore, is the largest Bethlehem plant, with an annual rated capacity of 6.2 million ingot tons. There is considerable diversity of product, but a high percentage is sheet, tin plate, galvanized sheets, and rod and wire. The large amount of light-weight end product requires a number of pickling installations that necessarily consume large quantities of sulfuric acid. In the coking operations sulfuric acid is also consumed in recovery of ammonia as ammonium sulfate and in treatment of recovered light oil derivatives. Total sulfuric acid consumption in the plant is currently approaching 10,000 tons per month, of which 55 to 60 pct is required for pickling.

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TP 4234B. Manuscript, Dec. 12, 1955. New York Meeting, February 1956.

In 1951 the acute sulfur shortage caused grave concern as to future sulfuric acid supply for the Sparrows Point plant. Raw materials for acid making were already at hand in the form of hydrogen sulfide and pyrites sufficient to produce nearly half the sulfuric acid required at that time. It was also known that the Bethlehem mining activities would produce additional pyrites in the future. Late in 1951 Bethlehem management decided to build a 250-ton sulfuric acid plant at Sparrows Point to utilize these raw materials. The plant was built and put in operation in May 1953. The steel plant has expanded considerably since 1951, and it is now necessary to operate the acid plant far over rated capacity in order to supply plant demand. Production of acid in 1954 amounted to 93,000 tons.

Raw Materials for Acid Making: Coking operations at Sparrows Point produce a residual fuel gas in large quantity which has a net heating value of 525 to 545 Btu. Prior to 1950, plant surplus coke oven gas was sold to the local utility company. In

The Transactions papers appearing in MINING ENGINEERING *during 1956 will subsequently appear in AIME Transactions Volume* 205, *and may be permanently referenced to that volume.*

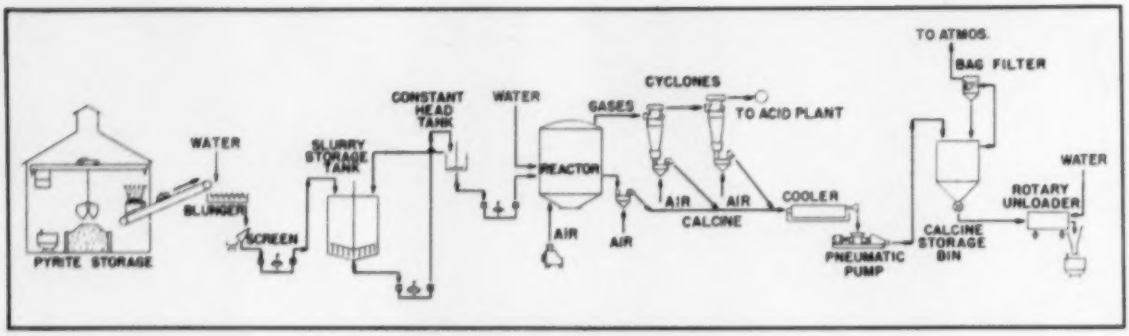


Fig. 1—Roasting flowsheet, Bethlehem Steel Co. Sparrows Point plant.

that year, however, a conversion was made from manufactured to natural gas in the area, and it became necessary to find additional usage in the plant for coke oven gas. This development had been known, and extensive testing indicated that substitution of coke oven gas and coal tar or pitch for fuel oil in the open hearth furnaces would actually reduce the heat time and consequently increase capacity of existing furnaces. Coke oven gas, however, contains hydrogen sulfide, the amount at Sparrows Point varying from 300 to 450 grains per 100 cu ft of gas. Since sulfur is undesirable in the open hearth furnaces, particularly if high capacity is required, it became necessary to remove a large percentage of the hydrogen sulfide from the coke oven gas that was to be used for open hearth fuel.

This also had been anticipated, and in July 1950, when the conversion to natural gas was made, the first unit of the coke oven gas desulfurizing plant went *on stream*. The unit was designed to process 20,000,000 cu ft of gas per day, reducing the hydrogen sulfide content to 50 grains per 100 cu ft.

As coking operations expanded, making available additional gas, and as the success of the new open hearth fuel was demonstrated, additional desulfurizing units were put in operation. No. 2 unit went *on stream* in 1952 and No. 3 in 1953. At the present time approximately 50 pct of the total coke oven gas, or 70 million cu ft per day, is being desulfurized and utilized in open hearth furnaces. This operation results in the production of 20 to 22 tons of hydrogen sulfide per day.

The hot vaccum actification process is employed to desulfurize the gas. This is a continuous liquid process where 2 pct sodium carbonate solution is employed to absorb hydrogen sulfide from the gas. The foul solution is continuously reactified with heat in a vacuum system. Hydrogen sulfide along with some hydrogen cyanide and carbon dioxide is continuously removed from the system. Hydrogen cyanide is undesirable in the acid plant, so it is removed from the mixture of acid gases by selective absorption in water. The resultant acid gas mixture, normally containing 90 pct hydrogen sulfide, is piped under negative pressure to a combustion chamber and waste-heat boiler at the acid plant.

Treatment of iron ore from Bethlehem's Cornwall mine near Lebanon, Pa., has resulted for a number of years in the production of a byproduct pyrite concentrate containing a recoverable quantity of cobalt. This pyrite was custom roasted for sulfur removal in multiple-hearth roasters and subsequently given a chloridizing roast prior to water leaching for the recovery of the cobalt. Work done by the Bethlehem raw materials research dept. indicated that it should be possible to sulfate roast the pyrite initially in a fluidized roaster and obtain higher cobalt recovery with elimination of the chloridizing roast. Gas from the sulfate roasting would contain 7.5 to 8.5 pct SO_2, which is sufficient for acid making. This has worked out satisfactorily in the plant. Approximately 50 to 55 tpd of Lebanon pyrite is currently roasted in one of the three reactors, producing a calcine that is water leached without further treatment. Cobalt recovery from the calcine averages nearly 90 pct.

Pyrites from Bethlehem's Grace mine near Morgantown, Pa., will eventually supply raw material for the other two reactors. They are, however, currently supplied with purchased pyrites obtained from Canadian sources. Current operating rate requires a 70-tpd feed to each of these reactors. The Canadian pyrite is dead burned, producing gas containing 12 pct SO_2 and calcine containing normally 0.25 to 0.35 pct total sulfur.

All calcine is sintered and utilized as part of the blast furnace burden.

Acid Plant General: Pyrites from the various sources are received in gondola cars and unloaded into separate bins in the pyrite storage building, which is equipped with an overhead crane. Canadian pyrite is shipped from April through November, which necessitates considerable stockpiling to carry through the winter. As bin storage is insufficient, it is necessary to truck some of the material from the building to outdoor storage. During the winter the pyrite is reclaimed from storage piles and trucked back to the bins as needed.

The roasting section of the plant consists of three fluidized bed reactors, each with its own auxiliary equipment. A detailed description of this section is presented below.

A blower located at the acid plant delivers the hydrogen sulfide through backfire preventers into the combustion chamber. Controls regulate the blower so that there is a slight suction in the hydrogen sulfide transmission line at the gas desulfurizing plant 4000 ft distant. Secondary air is controlled to the combustion chamber to reduce the temperature of the combustion gases to 1050°C. The gas containing approximately 7 pct SO_2 passes through a waste-heat boiler where it is cooled to 250°C, producing 7000 lb of steam per hr at 225 psig.

Gas from the waste heat boiler joins the gas from the three reactors in a common duct. The combined gas passes through a wet scrubber, where it is humidified and cooled to 35° to 37°C. Solids are reduced from 4.1 to 0.03 grains per cu ft. The gas next passes through two electrical precipitators in parallel where mist and additional solids are removed. The cool clean gas now enters the contact plant,

which is conventional in design. Vanadium pentoxide catalyst is employed in a three-stage converter.

Product acid is withdrawn from the 66°Be drying circuit and passes through a stripper where a stream of air removes free SO_3. The air and SO_3 are introduced into the main gas stream ahead of the drying tower.

Product acid, free of SO_3, enters one of two 800-ton running tanks. Acid for coke oven operations is pumped from these tanks directly to small storage stations at the two ammonium sulfate plants and the light-oil refining plant. Acid for pickling is transferred to the various mill storage stations in 10,000-gal tank cars. There are 11 of these cars in service. A central storage station consisting of two 3000-ton tanks permits a moderate working inventory.

The acid plant is operated as a part of the coal chemical section of the coke oven dept. On the day shift six maintenance men work five days a week and two work Saturday and Sunday. The day shift force includes: acid plant foreman, slurryman, car positioner, pyrite craneman, gas tester, and acid pumpman. All shifts include an acid plant turn foreman, contact operator, reactor operator, and two plant helpers. On the second and third shifts, one man works seven days per week. Additional maintenance men are drawn as required from general departmental or general plant forces.

Reactor Installation: The reactor installation consists of three 18-ft fluidized bed reactors. Each reactor is part of a completely independent unit. Reactors No. 1 and 2 burn at the present time a mixture of Canadian pyrites. Reactor No. 3 treats the cobalt-bearing Lebanon pyrite concentrate. The flowsheet of the roasting plant is shown in Fig. 1 and the general layout in Fig. 2.

The pyrite storage, pyrite slurry system, dust collecting system, calcine handling, and calcine storage and auxiliary equipment for each of the three reactors are identical.

Pyrite concentrate, containing 5 to 12 pct moisture, is loaded once a day into pan feeders 72-in. wide, provided with steel hoppers 5 ft 6 in. by 25x7 ft 10 in. high. From here the concentrate discharges

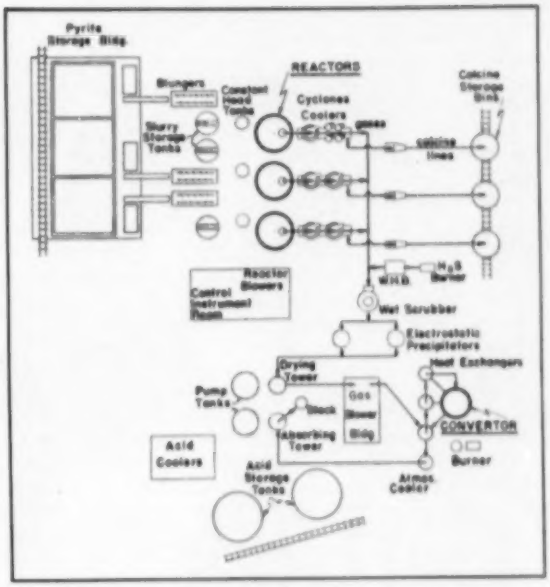
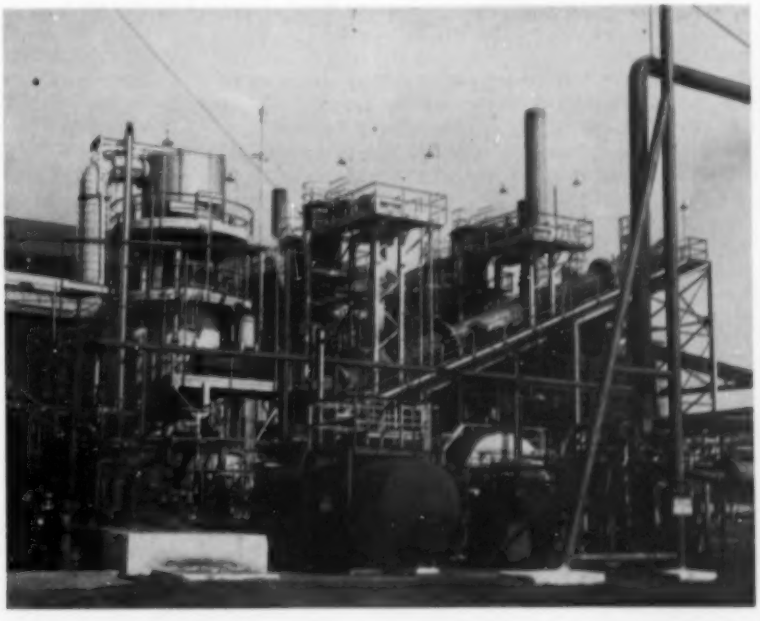


Fig. 2—General layout, reactor-sulfuric acid plant, Sparrows Point.

onto 24-in. inclined belt conveyors that transfer the concentrate into 5x12-ft double-shaft log washers or blungers. An overhead electromagnet removes iron scrap from the inclined belts. Water is added to the blungers to repulp the concentrate and prepare a slurry containing 74 to 78 pct solids.

Slurry from the blungers is screened on single-deck 3x6-ft low-head vibrating screens using ¼-in. mesh stainless steel cloth to remove trash. The slurry is pumped from a sump tank by 3-in. diaphragm slurry pumps into slurry-agitator tanks 12 ft diam and 14 ft high. Sufficient slurry is prepared and stored in the first shift to last for 24-hr operation.

Density of the pyrite slurry is checked manually in pulp density scales before it enters the slurry



Wet scrubber is shown to left of reactors at Sparrows Point plant. Common gas duct for three reactors is shown at center.

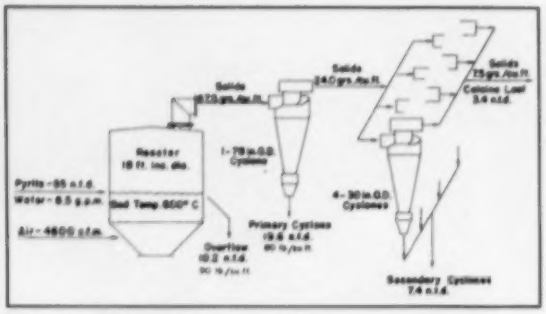


Fig. 3a—Pyrite dead roasting, dust loading reactor plant. Solids in grains per cubic foot dry gas measured at 15°C.

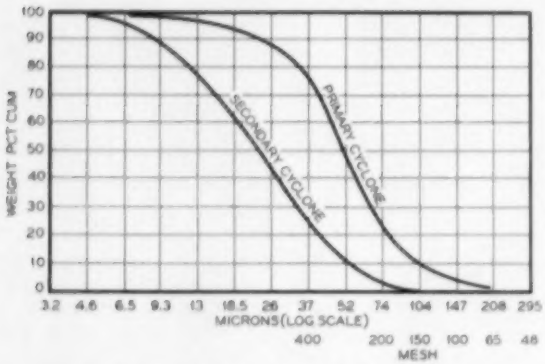


Fig. 3b—Screen analyses for pyrite dead roasting plant.

storage tanks. The density readings are used to control the amount of water added to the blunger.

Slurry from the slurry storage tanks is pumped by 2-in. diaphragm pumps into constant-head tanks 4 ft diam by 5 ft high. Excess slurry flows by gravity back to the slurry storage tank. From the constant-head tanks the pyrite slurry is finally pumped by 1½-in. diaphragm pumps into the reactors. Duplicate pumps insure continuous operation.

The abrasive character of the pyrite, together with the corrosive properties of the slurry, due to the presence of 2 to 15 pct pyrrhotite in the concentrates used, made it necessary to protect all the equipment handling the pyrite slurry against both abrasion and corrosion. Rubber covering and 316 stainless steel are used in the slurry-feed systems. Use of 316 stainless steel has greatly helped in maintaining a continuous trouble-free operation.

The roasters are single-compartment fluidized bed reactors. The 20 ft 3-in. diam steel shell is protected inside with 9 in. of insulating refractory and 4½ in. of fire brick and outside with 2¼ in. of insulation. The purpose of the insulation is to maintain skin temperatures of about 550°F to protect the steel against possible attack by the roasting atmosphere. A 5-ft fluid bed, composed of roasted pyrite particles, practically all coarser than 200 mesh (74 μ), is maintained inside the reactor by means of an overflow pipe. The freeboard amounts to about 10 ft.

The air to fluidize and roast the pyrite concentrate is supplied by multistage turbo blowers rated at 6000 scfm and 5 psig differential pressure. Air at about 4 psig enters the wind box of the reactor and is distributed throughout the area of the roaster by 332 nonsifting nozzles. A superficial velocity of 1.2 to 1.5 fps, measured at the roasting temperature, is maintained inside the reactor.

An auxiliary fuel oil burner is used to bring the fluid bed above the ignition temperature of the pyrite at the start of the roasting operation. The exothermic character of the chemical reaction resulting from the burning of the pyrite maintains the combustion process once it is started.

Pyrite slurry is fed into the reactor through a 1-in. diam stainless steel gun at about the level of the fluid bed. The gun extends about 12 in. inside the reactor wall. Each reactor is equipped with two guns.

The temperature to be maintained in the roasting operation is controlled by injection of water through the feed gun. A temperature-recorder-controller instrument regulates automatically an air-operated valve in the water line. The sensing element that activates the controller is a thermocouple that measures the bed temperature at a point 1 ft above the air-distributing bottom plate and below the feed gun.

Practically all the roasted pyrite particles smaller than 150 mesh (104 μ) leave the reactor with the gases resulting from the roasting operation. The carryover is roughly about 95 pct for reactors No. 1 and 2 and about 75 pct for reactor No. 3. Part of the solids (dust) is recovered in a two-stage cyclone system; the balance is collected by a wet scrubber. No attempt to recover the solids from the scrubber effluent has been made up to the present time. Loss of solids in the scrubber effluent amounts to about 8 pct of the weight of the feed for reactors No. 1 and 2 and about 12 pct of the weight of the feed for reactor No. 3.

The dust-collecting cyclone system consists of a single 78-in. diam cyclone per stage. Cyclones are protected inside with a 3-in. thick castable refractory lining and on the outside with 2¾ in. of block insulation.

Dust loss is particularly serious in the No. 3 unit, which handles the cobalt-bearing Lebanon pyrite concentrate. To improve the dust collecting efficiency in this unit, the secondary cyclone in the system described above has been replaced with a bank of four 30-in. diam stainless steel cyclones connected in parallel. With this arrangement the calcine loss has been decreased to 6 pct of the weight of the feed.

The poor collecting efficiency shown by the cyclones is not only due to the particle size of the calcine lost (50 to 55 pct of this material is smaller than 10 μ and practically all of it is —40 μ), but also to the character of the —60 μ calcine particles that often plug up the cyclone discharge legs. Use of thermocouples located in the seals of the discharge legs has helped warn the operator when the cyclones are not functioning properly. Usually, *rodding* or *blowing* a discharge leg is sufficient to open it.

The reactor overflow and each one of the cyclone discharge legs are equipped with *U*-type seals. One side of the *U* is connected to the cyclone tailpipe. The collected calcine is fluidized with a small amount of air (from 1 to 7 cfm per seal) and forms a barrier to the gaseous roasting atmosphere. As calcine collects in the seal, the excess solids overflow through the opposite side of the *U*.

These *U* seals also provide a convenient way to discharge the hot calcine (about 400°C) into rotary coolers 3 ft diam by 25 ft long that are partially submerged in water. The calcine is cooled here to about 40°C. From the cooler 3-in. pneumatic conveyor sys-

tems transfer the calcine into 150-ton storage bins. Rotary unloaders receive calcine from the storage bins, add about 12 pct water and discharge the moistened material into railroad cars. Addition of water is necessary to prevent *dusting* of the calcine.

Finally, a dual-bag filter unit on top of No. 3 bin is used to clean the air from all three pneumatic conveyor systems before it is exhausted to the atmosphere.

Instrumentation and Control: Temperatures in each of the reactors are measured and recorded at the wind box: at 1 and 3 ft above the bottom air-distributing plate, in the fluid bed; at 1.0 ft below the top of the reactor, in the freeboard; at the gas outlet from the secondary cyclone; at the discharge legs from the cyclones; at the cooler inlet and cooler discharge. The thermocouple which measures the bed temperature at 1.0 ft above the bottom plate is connected to a temperature-recorder-controller instrument. The water added to control the roasting temperature is regulated by this instrument. An indicating rotameter in the instrument panel measures the addition of cooling water. The operator logs the temperatures and the rate of cooling water every 2 hr.

Pressures at the wind box, bottom of the fluid bed, freeboard, and exhaust duct at the secondary cyclone are measured by indicating pressure gages located in the instrument panel. Air flow is measured by an orifice plate and the volume is recorded by instrument. Air flow is regulated manually by means of a butterfly valve. Pressures and air flow readings are logged every 2 hr.

The pyrite-slurry density is measured manually in pulp-density scales every 2 hr. The slurry sample is cut at the inlet of the constant-head tanks. Consumption of slurry is checked by measuring the drop of the slurry level in the slurry-agitator tanks at 2-hr intervals. The timer setting and the stroke length of the diaphragm feed pump are checked every 2 hr. All these readings are recorded on the operating log sheet. The amount of slurry fed is controlled by adjusting the number of strokes, the length of the stroke or both, in the diaphragm feeding pumps.

Composition of gases leaving the secondary cyclones is determined manually at least once a day. A continuous oxygen gas analyzer is used to record the oxygen content in the exhaust gases of any one of the three reactors. The gas analysis serves to check both the feeding and fluidizing air rates.

In practice the operator is given the amount of air, the tonnage, and the roasting temperature desired. The SO_2 and O_2 content of the exhaust gases indicates when the roasting is progressing properly. Once the amount of fluidizing air is fixed, it is a matter of adjusting the pyrite feed rate to maintain a given set of roasting conditions.

Composition, which is fairly constant, and tonnage of the pyrite concentrates processed in the reactor section are checked by sampling and weighing each railroad car received. Each composite of every source is sized weekly and analyzed for sulfur, iron, silica, copper, and zinc; Lebanon pyrite is also analyzed for cobalt. Because of the value of Lebanon pyrite, this concentrate is sampled again in slurry form at the constant-head tank. A weekly composite of samples cut every 2 hr is analyzed for sulfur, iron, cobalt, and copper.

Calcine output is sampled and weighed as it leaves the plant. Composition of the Canadian calcine, it was found, remains fairly constant, and weekly samples taken from the railroad cars are screen sized and analyzed for moisture, iron, sulfur, and silica. The Lebanon calcine is sampled every 2 hr at the cooler discharge and also at the railroad cars before it leaves the plant. The 2-hr sample is composited daily and the daily composites are recomposited weekly. Part of the daily sample is leached in water to determine the soluble cobalt content of the calcine. The cobalt extraction is computed from the cobalt content of the leaching solution. The weekly composite calcine is also leached in water, but the calcine, the leaching solution, and the leached residue are analyzed: the solids for iron, cobalt and sulfur, the solution for iron and cobalt. The cobalt content of the weekly composite calcine is used to compute the extraction in the daily composite. The calcine sample from the railroad cars is leached in the same manner.

The quantity and composition of the solids leaving the secondary cyclones are checked by samples taken from the water scrubber effluent. A sample is cut every 2 hr and composited daily. Consistency of daily results made possible a weekly recomposite.

The composite effluent sample is first analyzed for percent solids. Iron, cobalt, and sulfur are determined in the solids, and iron, cobalt, and sulfate radical in the clear solution. These analyses, together with the amount of water used in the scrubber, give a check on the dust losses from the reactor units. The water used in the scrubber is measured by rotameters and every 2 hr the readings are recorded in the scrubber daily report log sheet.

Dead Roasting of Pyrite: Pyrite concentrate is dead roasted in the fluidized bed reactors at Sparrows Point to convert the sulfur into sulfur dioxide gas of sufficient strength for manufacture of sulfuric acid. Secondary to the roasting operation is recovery of

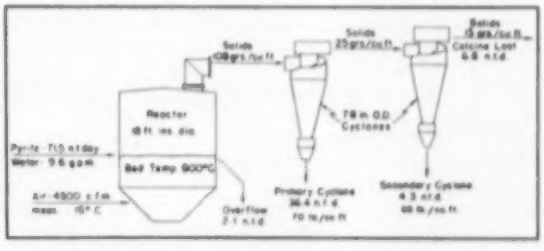


Fig. 4a—Pyrite sulfate roasting dust loading reactor plant. Solids in grain per cubic foot dry gas measured at 15°C.

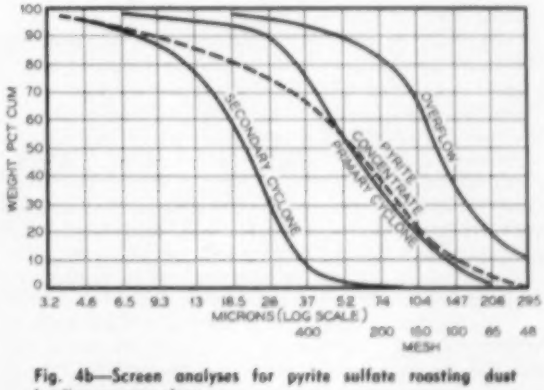


Fig. 4b—Screen analyses for pyrite sulfate roasting dust loading reactor plant.

Table I. Operating Data for Dead Roasting Pyrites

Weight of feed, pct: S, 46 to 50
Fe, 44 to 45
Cu, 0.06 to 0.40
Zn, nil to 0.90
SiO_2, 0.80 to 2.50

Sieve Analysis

Mesh	Cumulative Pct Retained
65	1 to 7
100	5 to 17
200	13 to 49
325	20 to 72

Roasting Operation

Feed: rate, 71.5 dry net tpd reactor
slurry, 78.0 pct solids
Air: 45 cu ft per lb, or 4500 cfm
Temperatures: wind box, 52°C
bed, bottom, 800°C
middle, 806°C
freeboard, 795°C
second cyclone, exit, 643°C
cooler inlet, 419°C
cooler outlet, 36°C
Pressures: wind box, 4.1 psig
fluid bed, 58 i.w.
freeboard, 13 i.w.
second cyclone, exit, 8 i.w.
Secondary cyclone exhaust gases, pct by volume:
SO_2, 11.5 (computed pct, 12.7)
SO_3, nil (computed pct, 0.7)*
O_2, 1.8 (computed pct, 2.8)
N_2, nil (computed pct, 83.8)
Moisture content: 29.0 pct (computed, wet basis)
Dust loading: 2.2 lb per 1000 cu ft (computed, dry basis)
Water to reactor: in slurry feed, 3.1 gpm per reactor
to control temperature, 6.5 gpm per reactor

Calcine Recovered, 43.8 Tpd Per Reactor

Weight, pct: Fe, 63 to 66
S, 0.15 to 0.50
SiO_2, 1.5 to 4.5

Sieve Analysis

Mesh	Cumulative Pct Retained
65	0.2 to 2.0
100	1 to 5
200	15 to 30
325	40 to 60

Calcine Lost (Computed), 6.6 Tpd Per Reactor

Fe, 56 to 57 pct
S, 2.0 to 2.5 pct

* Ratio of SO_2 to SO_3 determined by test.

Table II. Operating Data for Sulfate Roasting Lebanon Pyrite Concentrate

Weight of feed, pct: S, 48.9
Fe, 41.2
Co, 1.42
Cu, 0.31
Insoluble, 4.5

Sieve Analysis

Mesh	Cumulative Pct Retained
48	0.6
65	1.9
100	9.1
200	40.5
325	61.1

Roasting Operation

Feed: rate, 55 dry net tpd reactor
slurry, 75 pct solids
additives, 10 lb soda ash per ton of concentrate
Air: 60 cu ft per lb, or 4600 cfm
Temperatures: wind box, 49°C
bed, bottom, 600°C
middle, 614°C
freeboard, 603°F
second cyclone, exit, 566°C (legs=450°C)
cooler inlet, 360°C
outlet, 49°C
Pressures: wind box, 3.8 psig
fluid bed, 67 i.w.
freeboard, 15 i.w.
second cyclone, exit, 9 i.w.
Secondary cyclone exhaust gases, pct by volume:
SO_2, 7.4 (computed pct, 8.3)
SO_3, nil (computed pct, 1.3)*
O_2, 7.8 (computed pct, 7.2)
N_2, nil (computed pct, 83.2)
Moisture content: 25.6 pct (computed, wet basis)
Dust loading: 1.09 lb per cu ft (computed, dry basis)
Water to reactor: in slurry pulp, 3.0 gpm
to control temperature, 5.5 gpm

Calcine Recovered, 37.2 Dry Net Tons Per Reactor

Weight, pct: Fe, 57.0; Co, 1.95; and S, 3.8.

Sieve Analysis

Mesh	Cumulative Pct Retained
65	1.1
100	5.1
200	30.6
325	52.0

Calcine Lost (Computed), 3.4 Dry Net Tons Per Reactor

Fe, 43.5 pct
Co, 1.75 pct
S, 11.2 pct
Cobalt solubility, 89.5 pct
Ratio of Co to Fe solution, 3.5 to 1.0

Material	Weight, Pct	Roasting Balance Analysis Fe, Pct	Co, Pct	S, Pct	Distribution Fe, Pct	Co, Pct	S, Pct
Concentrate:	100.0	41.2	1.42	48.9	100.0	100.0	100.0
Calcine							
Overflow	18.5	58.6	1.84	2.3	26.2	23.9	0.9
First cyclone	35.6	61.2	2.00	1.7	52.9	52.0	1.2
Second cyclone	13.5	43.5	1.75	11.2	14.2	16.5	3.1
Composite	67.6	57.0	1.95	3.8	93.4	92.4	5.2
Losses	6.2	43.7	1.74	11.4	6.6	7.6	1.4
Total	73.8	55.8	1.93	4.4	100.0	100.0	6.6

Gases,** Pct by Volume: SO_2, 8.3; SO_3, 1.3; O_2, 7.3; N_2, 83.2.

Material	Weight, Pct	Leaching Balance Analysis Fe, Pct	Co, Pct	S, Pct	Distribution Fe, Pct	Co, Pct	S, Pct
Leached Residues							
Overflow	16.5	65.5	0.30	0.23	26.2	2.3	0.08
First cyclone	32.3	67.1	0.27	0.09	52.6	6.1	0.06
Second cyclone	9.4	60.5	0.29	0.37	13.8	1.9	0.07
Composite	58.2	65.6	0.25	0.17	92.6	10.3	0.21
Solutions							
Overflow	Co to Fe	6.6			0.11	21.6	0.80
First cyclone	Co to Fe	8.1			0.20	45.9	1.18
Second cyclone	Co to Fe	1.1			0.47	14.6	3.06
Composite	Co to Fe	3.6			0.78	82.1	5.04

** 81.0 pct of the sulphur content of the gases was in the form of SO_2 and 12.4 pct in the form of SO_3.

the iron content of the concentrate as an iron oxide calcine.

Roasting conditions necessary to insure the maximum generation of sulfur dioxide gas are known. In the dead roasting operation at Sparrows Point, 93 to 95 pct of the sulfur in the pyrite is converted to a sulfur dioxide gas analyzing 12 to 13 pct SO_2. The oxygen content of these gases is maintained at 1 to 2 pct. Typical operating data obtained during the dead roasting pyrite concentrates are summarized in Table I, and dust loadings in the gaseous reactor products are shown in Figs. 3a and 3b.

The fluidized bed reactors have been operated at different rates of feed and with four different kinds of pyrite concentrates. Feed rates have varied from a low of 35 tpd up to 90 tpd. At the highest feed rate the calcine conveying equipment reached its maximum capacity. One of the pyrite concentrates was so fine that it was necessary to operate the reactor with no overflow, and flue dust was mixed with the concentrate to maintain a satisfactory bed level. The level of the fluid bed is checked by measuring the pressure at the bottom of fluid bed; 60 to 65 i.w. are maintained in all three reactors.

The sulfuric acid section of the plant is shut down at least once a year for about a week. In these instances the reactors are blanked off. The fluid beds cool down at a rate of 50° to 60°C per day in moder-

ate weather. If the shutdown does not last more than 7 days the reactors are put into operation without preheating the beds.

The overall conversion of sulfur into sulfuric acid, obtained by dead roasting pyrite concentrates, is 88 to 90 pct of the sulfide sulfur.

Forty-four thousand dry net tons of Canadian pyrite concentrate were dead roasted in 1954. Twenty-five thousand three hundred dry net tons of calcine were recovered. The sulfuric acid manufactured from these pyrites was estimated at 58,200 net tons of 100 pct sulfuric acid.

Cobalt Sulfate Roasting: Roasting conditions that promote sulfatization are the opposite of those maintained in dead roasting. The theory of sulfate roasting is well known. Commerical application of these conditions to both preferentially sulfate cobalt and production of sulfur dioxide gas of sufficient strength to manufacture sulfuric acid has not been done before in a single roasting operation. Typical operating data obtained during the sulfate roasting of Lebanon concentrates are summarized in Table II, and dust loadings in the gaseous reactor products are shown in Figs. 4a and 4b.

The ratio of iron to cobalt in the Lebanon pyrite concentrate varies from 20 Fe to 1 Co to 30 Fe to 1 Co. The reactor roasting of this concentrate under conditions to promote the preferential sulfatization of the cobalt has given an average 90 pct cobalt extraction at the leaching plant. Average iron solubility is about 0.75 pct of the iron contained in the sulfated calcine. The ratio of cobalt to iron in the leaching solution averages 3.5 Co to 1 Fe.

The sulfur dioxide content in the gases leaving the secondary cyclone is 7 to 8 pct, the oxygen content 6 to 7 pct. Conversion of sulfur into sulfur dioxide, obtained by sulfate roasting, is 80 to 83 pct of the sulfide sulfur. Overall conversion of sulfur to sulfuric acid is 75 to 78 pct of the sulfide sulfur.

In the course of investigating the effect of the sulfate roasting gases on production of sulfuric acid, Lebanon pyrite concentrate was dead roasted for five months in 1954. The concentrate was roasted at 1560°F (850°C). A feed rate of 50 to 55 tpd with an air rate of 3500 to 3600 cfm (46 cu ft per lb of concentrate) was maintained. Cobalt solubility dropped to about 0.7 pct, sulfur conversion to sulfur dioxide gas increased to 86 to 88 pct, and overall conversion of sulfur to sulfuric acid increased to 81 to 83 pct of the sulfide sulfur. The dead burned calcine was chloridize roasted and cobalt extractions of 90 to 92 pct were obtained. This cobalt extraction was much higher than that obtained when the same material had been dead burned in previous years in multiple-hearth roasters.

To date Lebanon pyrite concentrate has been sulfate roasted at rates of 35 to 55 tpd. Availability of this concentrate has dictated the roasting rate. From August to December 1954 a total of 5600 dry net tons were sulfated. Calcine shipped in the same period of time totaled 3500 dry net tons. Production of sulfuric acid from the sulfate roasting operation was estimated at 6700 net tons of 100 pct sulfuric acid.

Operational and Maintenance Difficulties: The original installation included centrifugal pumps for slurry handling. Such pumps had performed satisfactorily on similar applications. In this case, however, the pyrite contained pyrrhotite and was much more abrasive than anticipated. The cast iron pumps were a continual source of trouble in the initial operating stage. In some cases rotors were replaced after 24 to 30 hr of service. A change to rubber-lined pneumatic pumps was quickly made. These pumps have proved satisfactory.

The severe abrasive and corrosive condition also made it necessary to change the slurry agitator mechanisms from steel to type 316 stainless steel.

There is considerable replacement required on blunger shafts and paddles as well as on the rotors and barrels of the calcine pumps that feed the pneumatic conveying system. Some change in material is called for here.

Piping in the calcine conveying lines, particularly the bends, has relatively short life. A number of experimental pieces have been installed, including hard cast iron-backed steel pipe and silicon carbide-lined pipe.

Originally the calcine from the cyclones was collected in water-jacketed stainless steel screw conveyors, which fed the material into the rotary coolers. These conveyors proved to be high maintenance items. When it was found that one of the three sections could be eliminated, this change was made on all three reactors, reducing screw maintenance by a third. Recently the shortened screw was completely eliminated on one reactor. Feeding calcine directly into the cooler appears satisfactory, and it is probable that the screws will be eliminated entirely.

Some difficulties have been encountered due to particle size of the pyrites. The original purchased pyrite was extremely fine (85 pct −325 mesh) and contained considerable pyrrhotite. Due to the fine particle size it was necessary to add flue dust to maintain a reactor bed, and of course a high percentage of calcine passed through the cyclones and was lost in the scrubber effluent. The material showed a tendency to pelletize in repulping, which resulted in a high rate of reject from the vibrating screen. This rejected material was recycled. There was also pelletizing in the slurry-circulating system which eventually caused stoppages. Pyrrhotite may have been responsible for the pelletizing.

Material from another source was considerably coarser with 40 pct −325 mesh. This pyrite slurried nicely and even when it was blended with 50 pct of the first material there was no difficulty in maintaining reactor beds at high operating rates. Using the first material alone it was impossible to operate over 50 pct of rated reactor capacity. Later shipments of the second material were still coarser, ranging from 25 to 35 pct −325 mesh. Reduction of fines to this extent proved unsatisfactory when the material was used without blending. In the Sparrows Point slurry system with the pyrites used it has been found essential that there be approximately 40 pct fines (−325 mesh); otherwise the slurry is not stable and the coarser particles settle out, causing stoppages.

During early operation of the plant there were several occasions where reactor beds became clinkered and defluidized. It was necessary to remove the clinkered material and make a new bed. This condition was largely the result of unsatisfactory operation of the slurry system, particularly the pumps. Overfeeding or overaddition of water for control usually started the trouble. Since operation of the auxiliary equipment has been improved there has been no difficulty with reactor operation. Since early 1954 plant delays due to the reactors have been practically nonexistent.

Discussion sent to AIME (2 copies) before Aug. 31, 1956 will appear in MINING ENGINEERING and in AIME *Transactions*, Vol. 205, 1956.

More Rock Per Dollar From the MacIntyre Pit

by F. R. Jones

AT Tahawus, N. Y., National Lead Co. operates the MacIntyre development. Here the world's largest titanium mine produces 5200 long tons of ore per day and pours 8000 long tons of waste rock over its dumps. Concentrated ilmenite is sent by rail to National Lead Co. pigment plants, and a second product, magnetite, is sold to steel producers in raw form or is agglomerated and shipped as sinter.

Several earlier attempts had been made to produce iron from the deposits, which have been known since 1826. These attempts failed, chiefly because of titanium impurity. In 1941 the present owners reestablished the operation for production of war-scarce ilmenite, and the impurity became the main product.

The Ore: The MacIntyre ore zone is about 2400 ft long and 800 ft wide in horizontal measurements. Ore outcrops were found on the northwest side of Sanford Hill, 450 ft above Sanford Lake and 2500 ft southeast. The zone dips at about 45° toward the lake and plunges to the southwest.

The ore minerals, ilmenite and magnetite, are unevenly distributed in bands roughly parallel to the long axis of the ore zone and are interspersed with bands and *horses* of waste. Hanging wall ores are fine grained and grade from rich ore to waste rock or gabbro. Footwall ores are coarse grained and are almost entirely ilmenite and magnetite. The footwall waste rock, anorthosite, is the common country rock. Several faults cut the ore zone. These faults have no great displacement but do contribute to the great physical variations in ore rock and surrounding waste.

The Mine: The MacIntyre mine is an open pit operation, with benches at 35-ft intervals. The lowest bench is now 54 ft below lake level.

Loading equipment consists of three electric-powered shovels (a P & H model 1400 with 4-yd dipper and two Bucyrus-Erie models 85-B with 2½-yd dippers) and one diesel-powered shovel (a Northwest model 80D with 2½-yd dipper).

Ore and waste are transported to a 48x60-in. jaw crusher in ten 22-ton Euclid trucks with 300-hp diesel engines. Ordinarily the two Bucyrus-Erie 2½-yd shovels load ore into a fleet of three or four trucks. This combination works two 8-hr shifts per day, moving 5200 long tons of ore to the crusher and removing a small portion of the waste rock. The P & H model 1400 shovel, with a fleet of four trucks, loads waste on three shifts per day. The mine operates on a 5-day week, with a small maintenance crew working Saturday.

Oversize rock is broken by a dropball handled by an Osgood model 825 rubber-mounted crane.[1] Ore and waste are broken by drilling and blasting 9-in. diam vertical holes behind the benches. Bucyrus-Erie 42-T churn drills are used to drill the holes, which are extended 4 ft below the bench level on which the broken rock will fall.

Drilling and Blasting History: In its early years the mine was equipped with Bucyrus-Erie 29-T churn drills, which drilled 6-in. holes. To keep up with production requirements the hole diameter was soon increased to 9 in., and by 1950 the three 42-T drills now in use had been acquired.

Early blasting experiments with different kinds and grades of explosive led to adoption of 90 pct straight gelatin dynamite as standard. It was recognized that this explosive was expensive, and from the start of operations until 1950 extensive experiments were made using blasting agents of the ammonium nitrate family. Results were recorded as uniformly poor, with great build-up of oversize rock. The expense of these experiments, and the discouraging results, caused the abandonment of any expectation of breaking MacIntyre rock with anything but 90 pct straight gelatin dynamite.

Further standardization led to 9-in. well drillhole spacings set at 16 ft in ore and 18 ft in waste, exceptions being permitted only for unusual conditions. The hole burdens were theoretically about 22 ft. Due to the extreme back-slope of bench faces, caused by blasting with heavy charges of dynamite, actual burdens were commonly well over 30 ft. Lack of precise control resulted in many holes having a burden as light as 15 ft.

General practice was to stem 6 or 7 ft of hole with magnetite concentrate, the amount of stemming being left to the discretion of the pit foreman. Usually all holes in a row were fired instantaneously with Primacord detonating fuse. Millisecond delays were

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TP 4228A. Manuscript, Nov. 14, 1955. New York Meeting, February 1956.

used for occasional multiple rows only, or for other unusual conditions.

Frequent increases in cost of dynamite and well drilling supplies were countered by improved techniques, so that primary drilling and blasting cost per long ton changed very little during the recent life of the operation up to and including 1953, see Table I.

Table I. Cost History of Well Drilling and Blasting*

Year	Production Ore and Waste, Long Tons	Comparative Well Drilling, Cost per Ton Produced	Comparative Well Drill Blasting, Cost per Ton Produced
1947	1,706,898	135	101
1948	3,499,226	121	101
1949	3,498,202	88	81
1950	2,625,217	88	94
1951	2,781,621	88	104
1952	2,235,288	91	94
1953	3,368,753	100	100

* Base year 1953 = 100.

Further Testing: In view of the increasingly widespread use of cheaper explosives for open pit blasting, it was decided that a cheaper blasting procedure must be found for the MacIntyre mine. Because of previous poor results from costly experiments, it was realized that further testing must be carried out with utmost caution. The following procedure was outlined:

1) Records would be set up to indicate in detail what results were being obtained from present drilling and blasting practices.

2) To assure uniformity, burdens on blastholes would be carefully controlled by survey.

3) By careful relating of dynamite charge to burden, spacing, and kind of rock being blasted, possible economies with the present explosive would be explored.

4) Explosives manufacturers would be asked to recommend cheaper blasting techniques.

5) The technique apparently best suited to MacIntyre conditions would be given fair trial, under guidance of the blasting experts of the explosives manufacturer.

6) If test results indicated substantial savings, a continuous program of cost reduction by improved techniques would be followed.

Establishment of Records: The following system of records was established:

1) The mine engineering dept. provided a *shot map* on a scale of 1 in.:50 ft for each shot, or group of holes to be fired. On the map the toe burden of each hole was shown (thickness of rock to be sheared at the bench elevation on which broken rock falls) and depths of holes were indicated.

2) The mine superintendent or his assistant made a *shot sheet* detailing the loading of holes. Once testing was well advanced a standard form was devised, and charting the shot sheet, Fig. 1, was turned over to the mine engineering dept.

3) For visual comparison when new explosives were introduced, shots were photographed and prints produced at a standard scale, approximately 1 in.:25 ft at crest-line. In general, shots were photographed at two stages: freshly shot and about half loaded out.

4) The number of oversize chunks was recorded for each shot.

Starting in September 1954, the outlined test procedure was applied. Early records pointed up the irregularity of toe burden achieved by current practices. Heavy crest damage from instantaneous blasting of heavy dynamite charges caused burdens on subsequent holes to be very heavy, while lack of precise survey control resulted in many very light burdens, Figs. 1 and 2. To reduce crest damage, or *back-break*, millisecond delay firing with primarcord connectors was introduced for all shots.

Examination indicated that good blasting results throughout the pit were being obtained from shots with standard spacings (16 and 18 ft) and burdens up to at least 26 ft. In order to provide a starting base, all burdens were then *pegged* at 26 ft. With instantaneous blasting, a burden as low as 26 ft was seldom uniformly attainable. In shots drilled behind a millisecond delay blast it was unusual for the crest condition to impose a burden in excess of 26 ft for any hole, Fig. 2.

Four-Month Results: At the end of January 1955 the results shown in Table II had been obtained.

The overall picture was most encouraging. Savings up to 33 pct in drilling cost per ton and 22 pct in blasting cost per ton were indicated. During the four-month period the apparent combined saving on 1,088,011 tons of ore and waste produced was $31,462, as compared to costs for the first nine months of 1954. This saving was realized in spite of a dynamite price increase of 6.8 pct at the beginning of the test period.

Although the testing program did not attain the maximum saving possible by carefully relating dynamite charges to burden, spacing, and kind of rock, it was recognized that further saving by reduction of dynamite consumption per ton of ore and waste blasted would be relatively small and would be countered by increasing shovel maintenance costs and reduced production. It was therefore decided that the time was appropriate for beginning experiments with cheaper explosives or blasting agents.

Table II. Comparative Well Drilling and Blasting Costs*

Period	Production, Ore and Waste, Long Tons	Comparative Well Drilling, Cost per Ton Produced	Comparative Well Drill Blasting, Cost per Ton Produced
1953	3,368,753	129	90
1954 (first nine months)	2,363,718	100	100
September 1954 to January 1955 (four months)	1,088,011	78	78
January 1955	278,644	67	83

* Base: first nine months of 1954 = 100.

Choice of Experimental Material: In the choice of experimental explosive material the following considerations were involved: proved performance, safety, water resistance, flexibility, and cost.

Previous experience pointed to the need for a strong explosive. It was thought that use of a slow or weak explosive would defeat possible savings by increasing shovel maintenance and secondary breaking. It was considered wise, therefore, to try a fast, powerful explosive well proved in the field.

It was recognized that members of the dynamite family are relatively sensitive to shock and abrasion. As evidenced by MacIntyre's record of 14 years without blasting accidents, with due care and rigid

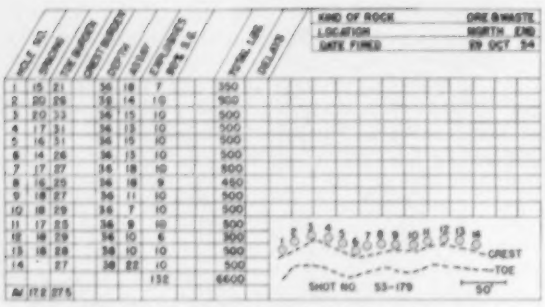

Fig. 1—Typical irregularity of toe burden behind instantaneous shots. Gabbro and anorthosite waste and ore. Average breakage. 0.305 lb per ton. Estimated tonnage, 21,600. 1540 tons per hole. Total chunks, 60.

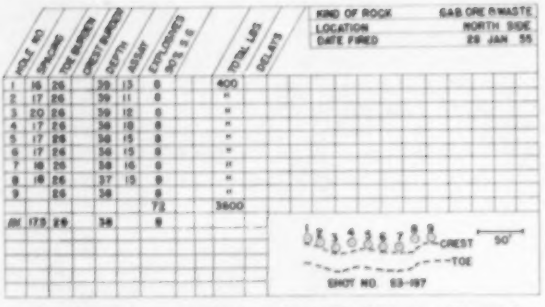

Fig. 2—Uniform toe burden obtainable behind millisecond delay shots. Apparently well broken. Half dozen oversize chunks. 0.237 lb per ton. Estimated tonnage 14,600. Truck tonnage, 15,200. 1700 tons per hole.

safety practice 90 pct straight gelatin dynamite can be handled safely. It was believed that the margin of safety could be increased by introducing a less sensitive explosive.

The MacIntyre pit is a compact unit. From a relatively small number of active benches, a variety of grades and types of ores are required for concentrator feed. Although shots are usually loaded and fired the same day, when the ore requirement is changed a shot is sometimes loaded and left for several days, and occasionally longer. Because ground water seepage and storm drainage often fill drillholes with water, water resistance was a prime factor in the choice of explosive.

Flexibility in load design and ease of loading and handling were also considered important.

Probable cost of explosive per ton of rock broken was the final criterion. Naturally, the cheapest material consistent with the above considerations was the most desirable.

Since manufacture of explosives and blasting agents is an intensely competitive industry, several manufacturers were able to recommend explosives that had an excellent chance of reducing MacIntyre blasting costs.

A line of *canned* explosives of the ammonium nitrate family was chosen as most likely to contribute maximum savings while fulfilling other requirements, Table III.

Testing Procedure: Under direct supervision of a technical expert provided by the explosive manufacturer, the first experimental shot using canned explosive *A*, pelletized explosive *B*, and canned primer charges was designed, with the same hole spacings and burdens standardized for recent dynamite shots, Table III and Fig. 3.

The experimental part of the pit was to be the advancing face of the new Sub-4 level, chosen because the ground was closely fractured and normally broke up well. The first shots were to be in waste rock, so that if poor results were obtained flow of ore to the crusher would not be hindered.

The first shot was fired Feb. 15, 1955, and appeared well broken both when freshly shot and when loaded out. The *powder factor* was held at a safely high figure, and the bottom load was increased by using 50 lb per hole of the free-running pelletized explosive *B*.

Other test shots in the above location were entirely satisfactory. Hole spacings and burdens were adjusted to a more nearly square pattern (burdens being reduced, spacings increased) reducing both drilling and explosive requirements per ton.

The new explosives were now tried in the more easily broken ground in other parts of the pit with equal success. By April 1, 1955, ten shots had been blasted, and a tabulation was made comparing results with those obtained by blasting during approximately the same period, and in similar ground, with 90 pct straight gelatin dynamite. A saving of 28.4 pct in blasting cost was shown, Table IV, with no apparent bad effect on any part of the operation.

Tougher Breaking: Sufficient experience had now been gained to consider extending use of the cheaper

Table III. Explosives Used In Experiments*

Product	Density	Sensitivity	Cartridge	Water Resistance		Price Per 100 Lb, $
				By Itself	In Container	
90 pct straight gelatin dynamite	32 lb per ft in 9-in. hole.	Cap-sensitive.	50 lb, 8-in. diam in waxed paper.	Excellent	Excellent	31.25
75 pct special gelatin dynamite	32 lb per ft in 9-in. hole.	Cap-sensitive.	As above.	Excellent	Excellent	24.65
Explosive *A* (Amytol, Sodatol, Soda-Amytol; cast)	41.7 lb per ft in 8½-in. diam can.	Not cap-sensitive.	Metal cans 2 ft long, various diam.	Poor	Excellent	17.70
Pelletized explosive *B* (TNT)	31 lb per ft in 9-in. hole.	Not cap-sensitive.	Free-running, used in bulk.	Excellent	———	20.00
Explosive *C* (Sensitized ammonium nitrate; packed)	37 lb per ft in 8½ in. can.	Not cap-sensitive.	Metal cans 2 ft long, various diam.	Poor	Excellent	16.64
Primer charge (Desensitized Amytol and grained Amytol; packed)	21 lb per ft in 7-in. diam cans.	Not cap-sensitive.	As above.	Poor	Excellent	16.64
Explosive *D* (Sensitized ammonium nitrate; packed)	37 lb per ft in 8½ in. diam carton.	Not cap-sensitive.	Cardboard carton with metal ends.	Poor	24 hr in 10 ft of water	10.50

* The term *explosive* is used in the general sense and includes blasting agents. Prices quoted, for 7-in. diam cans and cartons or larger, are those in effect during the period covered by this discussion.

Table IV. Experimental Shots Compared to 90 Pct S. G. Shots in Similar Rock Jan. 1 to April 1, 1955

Shots Fired	Long Tons Broken	Oversize Chunks Per 1000 Tons Broken	Explosive, Lb						
			90 Pct S.G.	75 Pct S.G.	A	C Plus Primer Charges	B	Total	Per Long Ton
18	343,400	3.82	74,100	2,150				76,250	0.222
10	174,000	4.18			37,098	9,019	2,650	48,767	0.280*

* Cost saving 28.4 pct due to cheaper material.

Table V. Early Results Experimental Blasting in Footwall Rock

Shots Fired	Long Tons Broken	Oversize Chunks Per 1000 Tons	Explosive, Lb						
			90 Pct S.G.	75 Pct Sp.G.	A	C Plus Primer Charges	D	Total	Per Long Ton
2	14,100	44.3	3,000	1,900				4,900	0.348
3	15,800	42.9			4,738	2,032	1,232	8,002	0.505*

* Cost saving 17 pct due to cheaper material.

explosives to the tougher parts of the pit, that is, in the footwall zone. Here the ore is very dense, being composed almost entirely of ilmenite and magnetite. The waste is anorthosite. Both ore and waste are coarsely fractured, and when blasted they tend to *roll out* in blocks weighing many tons. Records indicated about four times as much oversize rock per thousand tons blasted from the footwall zones as from the hanging wall zones.

The first shot in footwall ore was fired May 27, 1955. Results were good, comparing well with the best dynamite shots in the same part of the pit. This shot and two subsequent shots were heavily loaded to insure breakage. Despite the extra heavy loading, a cost margin of 17 pct was maintained over two immediately prior shots using dynamite in adjacent and similar rock, Table V.

Extended Use of Cheaper Explosives: The successful use of cheaper explosives in the tougher breaking zones was extended. It was now indicated that the cost savings being made in drilling and blasting were not reappearing in other pit costs, Table VI.

The addition of explosive *D* in May 1955 to the list of available blasting agents permitted further reductions in blasting costs, Table VII. Although MacIntyre was still using a few thousand pounds of 90 pct S.G. dynamite monthly, in order to use up material stocked by local suppliers especially for the MacIntyre operation, the cost per long ton for blasting from May 1 to Aug. 1, 1955, was reduced by 47 pct as compared with 1954. It is anticipated that there will be further reduction when local stocks of dynamite are exhausted.

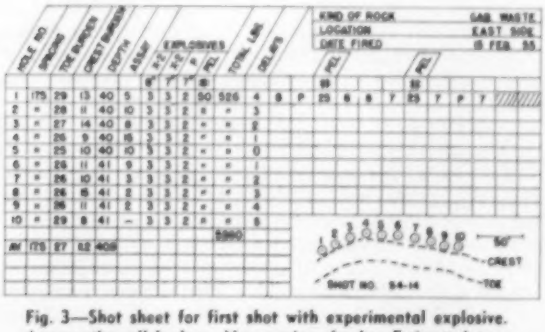

Fig. 3—Shot sheet for first shot with experimental explosive. Apparently well broken. No oversize chunks. Estimated tonnage, 17,200. Truck tonnage, 18,300. 0.287 lb per ton.

Combined Saving, Drilling and Blasting: Indicated reduction in well drilling cost since the start of the present program is 28.9 pct compared to costs for the first nine months of 1954. Reduction in well drill blasting cost over the same period is 50.0 pct.

Table VI. Mining Cost Per Long Ton Produced*

Period	Comparative Cost
1954 (first nine months)	100
October 1954 to January 1955 (four months)	100
1955 (first eight months)	85

* Base: first nine months of 1954 = 100.

Combined saving on the above basis at the present rate of production is $188,000 per year. Compared to 1953 costs, the saving is $200,000 per year.

Conclusion

With the foregoing experimental program, ground once proved too tough to break by an explosive other than 90 pct straight gelatin dynamite is now being satisfactorily blasted by other explosives for half the cost. Drilling costs were reduced nearly one third during the course of the same program, without any improvement in drilling equipment, or even in drilling speed.

It is the author's opinion that in any open pit operation where the equivalent of the foregoing experimental program has not been carried out, such a program will produce more rock per dollar.

Blasting Technique Development: In the first of the experimental shots with canned explosive *A*, canned primer charges, and pelletized explosive *B*, it was thought desirable to initiate all shots at the bottoms of the holes, i.e., point of maximum confinement. To get this result the primacord line to the top primer was delayed 9 milliseconds behind the line to the bottom primer. Later shots were fired by simultaneous firing of both lines of primacord, with no apparent disadvantage.

In the first half dozen experimental shots, 8-in. diam cans of explosive *A* were used as a bottom charge, augmented by the use of pelletized explosive *B*. When 8½-in. cans of explosive *A* were obtained the pelletized explosive, $2 more per 100 lb, was eliminated except where there was a heavier than normal burden.

Table VII. Comparative Costs of Well Drilling and Blasting, Showing Reductions to Aug. 31, 1955*

Period	Production Ore and Waste, Long Tons	Comparative Well Drilling, Cost Per Long Ton Produced	Comparative Well Drill Blasting, Cost Per Long Ton Produced
1954	3,173,085	100	100
1955 (first eight months)	3,224,120	75	65
May 1 to Aug. 31, 1955	1,169,947	75	53

* Base: 1954 = 100.

A few shots were fired using a *decked* charge of a single canned primer, but this means of stringing out a charge was discontinued, and a continuous column of 6-in. cans of explosive *C* was substituted.

Soon after the publicity attending the development of *Akremite*, explosive *D* was made available. This cheaper and bulkier material is currently being used in 6-in. and 7½-in. diam sizes to string out loads.

The former standard practice of stemming shots 6 or 7 ft above the top of the explosive column was abandoned in favor of stemming to the hole collar. With this later practice, spouting of holes caused by stemming blown out of the hole is uncommon, indicating full utilization of gases released by explosion.

Acknowledgments

The author extends thanks to National Lead Co. for permission to publish information in this paper; to representatives of all powder suppliers, especially Bart Carr and Charles Pope, of E. I. du Pont de Nemours & Co. Inc., who helped load experimental shots in the bitterest of winter weather; to the MacIntyre staff; and to the MacIntyre mine crew, without whose help these experiments could not have been carried out.

Reference

[1] P. W. Allen: Dropball Cuts Blasting Costs at Tahawus. MINING ENGINEERING, April 1953, p. 379.

Discussion sent (2 copies) to AIME by Aug. 31, 1956, will appear in MINING ENGINEERING and in AIME *Transactions*, Vol. 205, 1956.

Processing and Marketing Muscovite Block And Film Mica

by Robert D. Thomson

LISTED among strategic and critical minerals for stockpiling, mica is an important raw material in electronic and electrical equipment industries. Each year large quantities are consumed in the manufacture of television and radio tubes, radar sets, generators, transformers, electric toasters, and irons.

Crude mica, which comes from the mine in crystals or in rough books of various shapes and sizes, is known as run-of-mine or book mica. Structural imperfections, impurities, rough edges, and defective laminae are removed by cobbing, rifting, and trimming to produce a marketable block mica or film mica.[1] After the rough books have been hand-cobbed to remove adhering rock, dirt, and crushed or defective mica they are rifted, or split. In the U.S. a straight-bladed knife is generally used for rifting. Products of this operation are classed broadly as untrimmed sheet mica and scrap.

Sheet mica is any relatively flat piece sufficiently free from physical defects to be cut or stamped into specified shapes for industrial use. Sheet mica is classed as *block*, *film*, and *splittings*. Block mica is not less than 0.007 in. thick with a minimum usable area of 1 sq in. Film mica is split from the better qualities of block mica to groups of specified thickness ranging from 0.0012 to 0.004 in. Splittings are sheets with maximum thickness of 0.0012 in. and minimum usable area of ¾ sq in. As virtually no mica splittings are produced in the U.S., they will not be discussed in this report.

Small, low quality block is subdivided as *punch* and *washer*. Punch mica is thumb-trimmed material large enough to yield a circular disk of at least 1½ in. diam for stained quality mica and 1¼ to 2 in. for clear quality. Poorer lots, often referred to as washer mica, are also included in the general term *punch* mica.

Trimming, the next step in processing sheet mica, removes broken and ragged edges, loose scales, and other major imperfections. Trimming may be done with a knife, sickle, shears, or fingers. In the U.S. it is usually done with a 3-in. linoleum knife to obtain a bevel-edged cut. If ragged edges are broken off with the thumb and fingers, the mica is called thumb-trimmed. Full-trimmed is rifted mica trimmed on all sides with all cracks, reeves, and

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TP 4227H. Manuscript, Sept. 22, 1955. New York Meeting, February 1956.

cross-grains removed. Half-trimmed is trimmed on two adjacent sides, with at least two thirds of the mica trimmed and no cracks extending into the usable area.

Large quantities of high quality, large sheet mica are ruined by improper rifting and by overtrimming. Experienced rifters and trimmers salvage punch and even block from material less experienced workers would discard. Regardless of methods used or precautions taken, however, there are sizable quantities of bench scrap.

Trimming produces block or film mica irregular in size and shape, which must be classified according to grade (size) and quality before marketing. No other mineral is so difficult to process, and none approaches the multiplicity of qualities and grades into which mica is grouped.

Grades are based on the maximum usable rectangle that can be cut or stamped from a piece. According to specifications of the American Society for Testing Materials, block and film mica are divided into 12 grades, ranging from No. 6, with a minimum area of 1 sq in., to the OOEE Special, with a minimum area of 100 sq in.[5]

Visual classification is based on the relative quantity of visible inclusions, such as air bubbles; stains caused by clay, iron and manganese compounds, or organic matter; waviness; and other structural imperfections. Clear mica, which is hard, uniform in color, nearly flat, free from all stains, inclusions, and other structural defects, is the highest quality. Block mica is divided into 11 quality groups, ranging from the poorest, *densely stained*, to the best, *clear*. Qualities of film mica are *first* (equivalent of the three highest qualities of block mica), *second* (equivalent of fourth quality block), and *third* (equivalent of fifth quality block). Any muscovite mica classified within the limits of the five highest qualities of block mica or the three film qualities is considered strategic.

Table I. Comparison Between Domestic Classification of Mica and ASTM Standards

Domestic	ASTM
No. 1	Clear, and clear and slightly stained (1st and 2nd highest qualities)
	Fair stained (3rd)
No. 2	Good stained (4th)
No. 2 inferior	Stained (5th and 6th)
	Heavy stained (7th)
No. 3 (electric or stained)	Black spotted (9th)
	Black stained (10th)

Although the *commercial quality system* is based on ASTM specifications, standards vary from one company to another. Much foreign block and film mica is sold to industry according to quality groups, which deviate somewhat from the nomenclature of ASTM, permitting more defects than are allowed by ASTM specifications.

Because the small domestic miner usually lacks the experience to appraise mica for quality in accordance with ASTM standards, it has been customary in the U.S. during normal times for the small miner to sell thumb-trimmed or half-trimmed block and to qualify an entire lot of mica in a rough manner. Domestic block mica is divided into two primary groups: clear and stained. Clear mica is subdivided as *No. 1*, *No. 2*, and *No. 2 Inferior*. Black-stained and black-spotted mica are known as *No. 3, electric*, or *stained*. An approximate comparison between domestic quality classification and ASTM standards is given in Table I.

A system of qualifying block and film mica for fixed mica-dielectric capacitors, based on a combination of the electrical and physical properties, is covered by ASTM D-748-52T.[6] Specifications provide for use of a *Q-meter* to determine the power factor and a spark-coil test to detect conducting stains. In the past, block and film mica have been classified by these methods only on special occasions.

Sheet mica also is classified according to color. Mica occurs in many colors, ranging from green through shades of green and brown to pinkish buff and drab. The Federal Geological Survey has arranged muscovite mica into seven main color categories and three commercial groups, as shown in Table II.[4]

Table II. Federal Geological Survey Color Categories for Mica

Principal Classifications	Commercial Groups
Pinkish buff and drab, Cinnamon brown, Brown	Ruby
Brownish olive, Yellowish olive	Rum
Yellowish green, Green	Green

The National Bureau of Standards has established two color groups: 1) *ruby* and 2) *nonruby*, which includes green or brownish mica, disregarding variations such as air or mineral inclusions and stained areas. Mica for the National Stockpile is purchased on the Bureau of Standards color classification.

Ruby mica is preferred because its *Q* factor is fairly constant and it splits easily. Nonruby mica tends to be brittle, and its *Q* factor varies considerably and often is undesirably high. Normally, nonruby is accepted for strategic uses only if it passes both visual and electrical tests. For ordinary electrical uses, color has little significance.

Marketing: In 1954 consumption of block and film mica cut or stamped to dimension by domestic fabricators totaled 3,219,157 lb, of which 97 pct was block mica.[5] Block mica, an insulator in electrical equipment, is also used in manufacture of transmitting and receiving tubes, high-temperature steam-gage glass, diaphragms, and compasses. In the electronic field most of the block mica is consumed as bridges and spacers in radio and television tubes. This mica is largely stained quality (fifth and sixth qualities) and in 1954 represented 78 pct of the block mica fabricated for tubes, or 44 pct of that fabricated for all uses. Lower qualities of block mica, including punch mica, are used mainly as electrical insulators in household appliances.

Film mica is used in capacitors that hold or store an electrical charge. Capacitors require high quality film mica; of the 97,255 lb used in 1954, 17 pct was first quality and 75 pct second quality.

In peacetime civilian consumption requires large quantities of the lower quality block and film mica, and demand for the higher qualities is not excessive. However, in times of emergency demand for high quality mica suitable for essential military applications increases tremendously. From all indications, block and film mica, especially of the higher qualities, will be in high demand for many years to come.

Buyers of sheet mica can be classed as commercial and Government. Commercial buyers are either mica fabricators or importers. The highly specialized fabricating industry consists of 1) firms that fabricate mica to supply the electronic and electrical industries and 2) firms that manufacture electronic or electrical equipment and fabricate for their own consumption. Although fabricators usually purchase both domestic and foreign mica, in the past few years only those in North Carolina and Virginia have purchased domestic mica. Major fabricating plants are located principally in electrical manufacturing centers outside the mica-producing regions. Eight fabricating companies are in New York, six in New Jersey, four each in Massachusetts and North Carolina, two in Illinois, and one each in Ohio, Pennsylvania, Rhode Island, and Virginia. Most of the mica importers are in New York City.

In 1952 the U. S. Government authorized, through General Services Administration (GSA), a purchasing program for domestic muscovite sheet mica to terminate June 30, 1955, later extended to June 30, 1957. Program *A* was established for the purchase of muscovite ruby and nonruby block and film mica and program *B* for ruby and nonruby hand-cobbed mica. Block and film mica are purchased in three quality groups: *good stained* or better, *stained*, and *heavy stained*, and in three grade groups, *Nos. 5½* and *6*, *Nos. 4* and *5*, and *No. 3* or larger. All good stained or better block and film must be full-trimmed, but stained and heavy stained are accepted as full-trimmed or half-trimmed. Hand-cobbed mica purchased under program *B* must yield at least 40.5 lb of full-trimmed block or film mica, stained or better qualities, per short ton.

Table III. Domestic Mica Purchasing Program (GSA) Prices per Lb, Ruby and Nonruby Block and Film Mica

Grades	Qualities				
	Full-trimmed			Half-trimmed	
	Good Stained or Better, $	Stained, $	Heavy Stained, $	Stained, $	Heavy Stained, $
Ruby					
No. 3 and larger	70.00	18.00	13.00	12.00	8.00
No. 4 and No. 5	40.00	8.00	6.00	5.00	4.00
No. 5½ and No 6	15.00	5.00	3.00	3.00	2.00
Nonruby					
No. 3 and larger	70.00	14.40	10.40	9.60	6.40
No. 4 and No. 5	40.00	6.40	4.80	4.00	3.20
No. 5½ and No. 6	15.00	4.00	2.40	2.40	1.60

Purchasing depots are located at Spruce Pine, N. C., Franklin, N. H., and Custer, S. D. Ruby and nonruby mica should be delivered f.o.b. to the nearest depot, each lot containing one or the other; mixed lots are not accepted. Any person wishing to participate in either program should notify the GSA regional director having jurisdiction over the nearest depot or the superintendent of the depot. The superintendent of the respective depot will notify the participant of a delivery date. Each mica delivery is inspected at the depot. Any mica not conforming to minimum requirements is rejected and all costs except inspection cost are borne by the participant. A price schedule for domestic mica purchased by the Government is given in Table III.

The Government also purchases muscovite block and film mica produced in foreign countries. Ruby and nonruby block are purchased in the full-trimmed condition, good stained or better quality, grade 6 or larger. Ruby block also is accepted in the stained quality under certain conditions with a proportionate quantity of ruby good stained or better qualities. Half-trimmed block is purchased only for ruby of qualities lower than good stained. Ruby and nonruby film are acceptable as full-trimmed, first and second qualities. Foreign mica usually is bought by the Government for delivery to a GSA warehouse through Federal Supply Service, 34 Exchange Place, Jersey City, N. J.

Prices: Prices offered by fabricators for domestic sheet mica range from a few cents to more than $10.00 per lb. Quoted prices are nominal, and actual selling prices are determined by direct negotiation between buyer and seller after agreement as to quality and grade of particular lots. Prices in the North Carolina district in 1955 are shown in Table IV. Range in price for each grade is caused by differences in quality, preparation, and exactness of grading.

Table IV. Prices for Various Grades and Qualities of Sheet Mica in the North Carolina District, April 1955[a]

Grade (Size)	Price per Lb, $	
	Clear Quality	Stained Quality
Washer mica	0.06 to 0.10	0.06 to 0.10
Punch mica	0.10 to 0.16	0.10 to 0.16
1½x2 in.	0.70 to 1.60	—
2x2 in.	1.10 to 1.60	0.70 to 1.10
2x3 in.	1.60 to 2.00	1.20 to 1.60
3x3 in.	1.80 to 2.30	1.40 to 1.80
3x4 in.	2.00 to 2.60	1.60 to 2.10
3x5 in.	2.60 to 3.00	1.80 to 2.70
4x6 in.	2.75 to 4.00	2.00 to 3.00
6 x8 in.	4.00 to 8.00	2.25 to 3.50
8x10 in.	6.00 to 13.00	3.00 to 4.00

[a] Personal communication with a commercial mica buyer.

Prices paid by the Government for domestic mica under program *A* are three to five times the market price. Payment for each lot of ruby and nonruby block and film mica containing 18 pct or more good stained or better quality is made on the price schedule shown in Table III. Payment for any lot containing less than 18 pct good stained or better quality is made according to the price schedule, with adjustments to account for proportions of lower quality mica.

Government prices for domestic ruby or nonruby hand-cobbed mica under program *B* are $600 per short ton and $540 per short ton, respectively, based on a yield of 16.2 lb of good stained or better qualities and 24.3 lb of stained quality block or film mica per short ton. If actual yield exceeds the preliminary estimate, the participant will be paid the difference, and if the yield is less than preliminary estimates, the amount of overpayment must be repaid to the Government.

References

[1] R. D. Thomson: Marketing Sheet Mica. Bureau of Mines *Inf. Circ.* 7729, 1955, 20 pp.

[2] American Society for Testing Materials: Tentative Specifications for Natural Muscovite Mica Based on Visual Quality. D-351-53T, 1953, 9 pp.

[3] American Society for Testing Materials: Tentative Specifications for Natural Block Mica and Mica Films Suitable for Use in Fixed Mica-Dielectric Capacitors. D-748-52T, 1952, 16 pp.

[4] R. H. Jahns and F. W. Lancaster: Physical Characteristics of Commercial Sheet Muscovite in the Southeastern United States. U. S. Geol. Survey *Prof. Paper 225*, 1950, p. 43.

[5] U. S. Bureau of Mines: Mica—Block and Film Mica Fabricated in the Second-Half of 1954. Mineral Market Report, *MMS No. 2401*, 1955, 11 pp.

Discussion sent to AIME by Aug. 31, 1956, will appear in MINING ENGINEERING and in AIME *Transactions*, Vol. 205, 1956.

Differential Infrared Spectra of Adsorbed Monolayers-n-Hexanethiol on Zn Minerals

The recently developed pellet technique to investigate infrared spectra of solids is a unique method of studying solid surfaces. By this procedure suspensions of solids distributed uniformly and reproducibly in a KI or KBr matrix may be prepared, resulting in differential infrared spectra in which the absorption bands associated with the surface only may be obtained. Adsorption of hexanethiol on ZnS or ZnO from either the gaseous or the aqueous phase result in splitting out of water from the mineral surface. In the case of willemite, Zn–O–Si bonds appear to be broken, causing an increase in concentration of surface hydroxyl groups.

by Edward M. Eyring and Milton E. Wadsworth

BETTER understanding of solid surfaces and their associated adsorption products is of both academic and practical value. The study of detergents and their behavior in cleaning surfaces is fundamentally related to the nature of the products adsorbed and their ability to compete for certain surfaces. Similarly in flotation, the nature of the adsorbate before and after adsorption is basic to clear understanding of the process.

The recently developed pellet technique[1, 2] employing infrared spectroscopy provides a powerful tool for the study of solid surfaces. Absorption bands, which are infrared-active, may be observed for adsorbed products. Also any changes in the surface substrate that occur as a result of adsorption may be detected by means of differential spectra.

In applying the pellet technique to solids of large surface area, French and his coworkers[3] have surveyed the adsorption of organic flocculants on clay minerals, oleic acid on fluorite, and ammonia on cracking catalysts. Recent work on application of differential infrared spectroscopy to the study of solid surfaces[4] has demonstrated that the difference between a sample and a reference can be obtained with double-beam instrumentation. In this way displacement of surface products, adsorption of new products, and differences in the substrate, if any, may be detected.

Experimental Procedure: Potassium iodide pellets were used in this study, prepared from recrystallized analytical reagent grade KI. Grinding was carried out in a Fisher mechanical agate mortar. Two-gram plates containing the solid under study were pressed in a special evacuated die constructed in the laboratory at the University of Utah.[5]

Concentration depended on the solid in question. Zinc sulfide and willemite are sufficiently transparent in the infrared region for concentrations as high as 20 mg of mineral in 2g of KI to be used. Zinc oxide is even more transparent, permitting concentrations as high as 60 mg in a 2-g plate. For nondifferential operation, however, much lower concentrations (2 to 10mg/2g KI) had to be used.

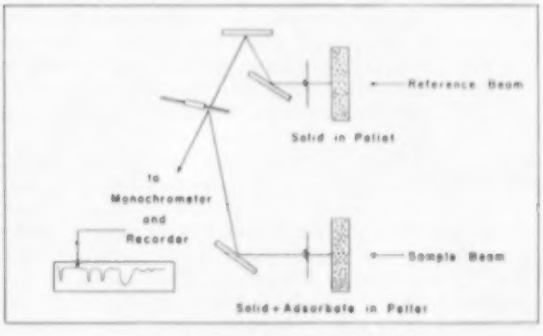

Fig. 1—Differential spectra of solids.

Spectra were obtained on a Perkin-Elmer 21 double-beam recording infrared spectrophotometer. A sodium chloride prism was used giving a spectral range of approximately 2 to 15 μ wave length. For differential runs a resolution of 1000 was used and the scanning speed was controlled commensurate with instrument response. The response was checked continuously and each spectrum was repeated several times. For nondifferential runs a resolution of 927 was used.

Zinc oxide used was analytical grade oxide fume. The zinc sulfide was precipitated from solution with analytical grade zinc sulfate and sodium sulfide. The precipitate was washed and centrifuged several times and dried. Before use the precipitate was washed in dilute HCl and then treated with saturated NH_4Cl to remove any bulk oxides and sulfates. X-ray patterns of the precipitate were comparable to that of the natural mineral sphalerite. The precipitate was much more satisfactory than finely ground natural ZnS because of its very large surface

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TP 4229B. Manuscript, Sept. 15, 1955. New York Meeting, February 1956.

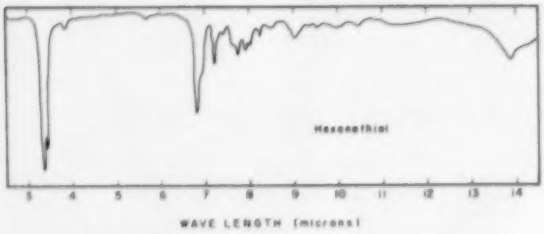


Fig. 2—Infrared spectrum of liquid n-hexanethiol.

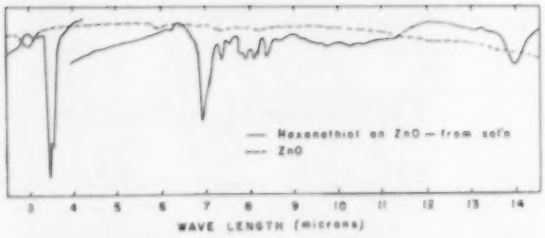


Fig. 3—Infrared spectrum of n-hexanethiol adsorbed on ZnO from aqueous solution.

area. The willemite (Zn_2SiO_4) used was the natural mineral from Franklin, N. J., carefully selected by hand under an ultraviolet lamp. Grinding was carried out for approximately 24 hr in a mechanical agate mortar, producing sufficient surface area to provide adequate differential spectra. Hexanethiol of analytical grade was supplied by the Eastman Kodak Co.

The advantage of double-beam instrumentation is that only the differences between the sample and the reference are recorded. Atmospheric CO_2 and H_2O are also cancelled out. In this procedure the sample beam passed through a pellet containing suspended mineral with hexanethiol adsorbed on the surface. The reference beam pellet contained the same mineral in the same concentration as the sample pellet, but no hexanethiol was adsorbed on the surface. In this way the spectrum of the mineral was cancelled out, leaving only the infrared spectrum of the surface. Fig. 1 is a diagrammatic sketch of the differential system.

Adsorption from the gas phase was carried out at room temperature in an evacuated dessicator by exposure for about 12 hr. The liquid hexanethiol was placed in a watch glass within the dessicator. Also present were the 2-g sample of mineral mixed with KI and a 2-g sample of pure KI. After exposure the dessicator was evacuated for several hours to remove physically adsorbed hexanethiol. This could be determined by examination of the KI by itself after pelletizing. This same procedure was followed for identical samples of KI plus mineral, and pure KI alone in the absence of hexanethiol. Each of the four samples, including the two plain KI samples, was pelletized under identical conditions. The sample beam pellets consisted of the KI plus mineral exposed to hexanethiol and the 2-g pellet of pure KI from the dessicator unexposed to hexanethiol. Reference beam pellets consisted of the mineral plus KI unexposed and the pure KI exposed to hexanethiol. In this way any mercaptan adsorbed on the KI by itself was cancelled out. The amount adsorbed on the KI was always very small but usually slightly detectable. Almost perfect compensation may be obtained by this procedure, since the sample and refence pellets were prepared from a common mixture of the mineral plus KI.

Adsorption from solution was carried out in stoppered erlenmeyer flasks (at neutral pH) containing the ground mineral plus an aqueous solution containing 80 mg per liter of hexanethiol. Contact times of less than 1 hr provided very adequate spectra. The mineral was washed with distilled water and filtered six times, after which it was dried under vacuum. Good differential compensation was somewhat more difficult to achieve, since the mineral had to be weighed separately for each pellet and could not be split from a common mixture. The mineral unexposed to hexanethiol was carried through the exact procedure of washing, filtering, and drying.

Discussion: This study was initiated as a result of the very informative work previously carried out on the same systems by Gaudin and Harris,[6] who used hexanethiol containing S^{35}. In this way the amount of hexanethiol adsorbed on zincite (ZnO), sphalerite (ZnS), and willemite (Zn_2SiO_4) could be directly determined by radioactive tracer techniques. Additional properties make these systems attractive to infrared studies: 1) the minerals are very transparent in the 2 to 15μ (NaCl) region of the spectrum, 2) they may be obtained or prepared with large surface areas, and 3) the hexanethiol has informative adsorption bands in the NaCl region. Fig. 2 is the infrared spectrum of the liquid hexanethiol. The deep double peak at 3.4 to 3.5μ is caused by C—H stretching frequencies associated with the methyl (CH_3) and the methylene (CH_2) groups. Of particular interest is the band at 3.95 μ. This band has been shown by several investigators[7] to be due to the —S—H stretching frequency of the mercaptan group. Although this band is very weak it may be readily detected for a spectra whose C—H stretching band is greater than 30 pct of total absorption. For the band to be detected on surfaces, therefore, the surface area must contain sufficient hexanethiol to exceed this 30 pct limit, a condition readily achieved with these mineral systems. In all cases used to establish the presence or absence of the S—H (3.95μ) stretching band, the C—H stretching band exceeded 60 pct of total absorption. The spectrum of liquid hexanethiol was obtained using a 0.012 mm liquid cell. The majority of other bands associated with the hexanethiol will not be discussed here since they were not directly affected by adsorption. The only exception that might be cited is the band at 13.7μ which is caused by the (CH_2), skeletal frequency. In the adsorbed state this band appears a little stronger and sharper. Such a difference might be expected between the same bands of a liquid and a solid, but in this instance the sharpening may be due to the close oriented packing of the mercaptan in the adsorbed state.

Results and Conclusions

Results of adsorption from the liquid and vapor phases respectively on ZnO are shown in Figs. 3 and 4. The dotted spectrum of Fig. 3 represents the spectrum of ZnO by itself. This is cancelled out by the differential method. Similar spectra were obtained for ZnS. Fig. 5 is the spectrum for hexanethiol adsorbed from liquid on ZnS. As is evident from the solid curve (differential spectrum), results were identical to those obtained for ZnO. Vapor phase adsorption on ZnS was also identical to that obtained for ZnO. These spectra were reproduced independently several times. The important features of these spectra are associated with the —OH stretching

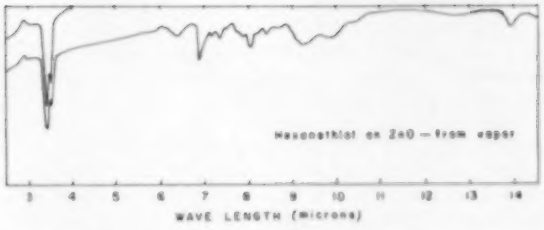


Fig. 4—Infrared spectrum of n-hexanethiol adsorbed on ZnO from vapor phase.

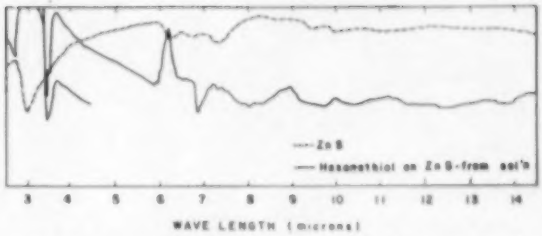
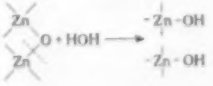


Fig. 5—Infrared spectrum of n-hexanethiol adsorbed on ZnS from aqueous solution.

(2.9μ) and bending (6.2μ) bands associated with surface hydroxyl groups on the mineral surfaces, the C—H stretching bands (3.40 to 3.5μ), and the —S—H stretching band at 3.95μ. Clearly in all these spectra the surface —S—H stretching (3.95μ) bands are missing. As indicated by the depth of the C—H stretching bands, concentration is sufficient to show the S—H band if it is present. Also common to all these spectra are the inverse peaks at 2.9μ (OH stretching) and 6.5μ (OH bending). The presence of bands going in the reverse direction (inverse) represents the bands of the reference pellet in excess of similar bands in the sample. These inverse bands clearly show that the surface has fewer surface —OH groups (therefore more in the reference beam) after adsorption of hexanethiol than before. Identical results were obtained for adsorption from either vapor or liquid phases.

Presence of surface —OH groups is readily detected by infrared spectroscopy. This is evident from either the spectra of the minerals by themselves or from the overcompensation of OH bands in the differential spectra. Their presence is probably associated with the satisfying of unsaturated valence forces when a surface is freshly fractured. This may be represented for zinc oxide by Eq. 1:



[1]

These —OH groups are undoubtedly present on the surfaces of most inorganic solids exposed to water either in the liquid or vapor phases. The surfaces encountered in either detergency or flotation problems appear to have in common, therefore, the presence of surface OH groups.

Vapor Phase Adsorption of Hexanethiol on ZnS and ZnO: Spectra obtained for ZnO and ZnS for vapor phase adsorption may be explained by the reaction between the undissociated hexanethiol in the gas phase and the surface according to Eq. 2:

$$-\overset{|}{Zn}-OH + HSR \longrightarrow -\overset{|}{Zn}-SR + H_2O \qquad [2]$$

During adsorption, water is formed, resulting in formation of the zinc salts as indicated in Eq. 2. This may be likened to the esterification that occurs during the reaction of organic alcohols and acids. The driving force appears to be predominantly the formation of water. Eq. 2 explains the spectra obtained. Formation of water results in removal of —OH groups from the surface of the mineral and at the same time the S—H group is broken, resulting in the surface zinc mercaptan bond. Removal of surface —OH causes over-compensation (inverse bands) at 2.9μ (—OH stretching) and 6.2μ (—OH bending) wave lengths. The —SH stretching bands disappear in the adsorbed state, also clearly evident in the spectra. Competition for surface sites therefore appears to exist between the hexanethiol and water.

Aqueous Phase Adsorption of ZnO and ZnS: Although identical spectra are obtained for both vapor phase and aqueous phase adsorption, alternate mechanisms may be proposed to explain the observed spectra. According to Wark[8] and Gaudin[9] the adsorption of collectors involves competition between the collector anion (in the case of anionic collectors) and the hydroxyl ion. This process is represented in Eq. 3 for ZnO and ZnS,

$$-\overset{|}{Zn}-OH + RS^- \rightleftharpoons -\overset{|}{Zn}-SR + OH^- . \qquad [3]$$

where RS^- represents concentration of the dissociated anion of hexanethiol. Such a reaction results in the removal of —OH groups from the surface. Also no —SH stretching band should appear. Both conditions are satisfied in the spectra obtained.

An alternate mechanism, also in agreement with the observed spectra, involves adsorption of undissociated hexanethiol from solution or ion pair adsorption as shown in Eq. 4. Large concentrations of the undissociate RSH are present in solution, since its dissociation constant is 1.9×10^{-11}.[10] Such a mechanism for the adsorption of collectors and depressants has been proposed by Cook et al.[11] There is, however, no direct criterion for choosing either of the above proposed mechanisms, since all three are consistent with the observed spectra.

$$-\overset{|}{Zn}-OH + H^+ + RS^- \rightleftharpoons -\overset{|}{Zn}-SR + H_2O \qquad [4]$$
$$\text{or}$$
$$(HSR)$$

Adsorption of Hexanethiol on Willemite (Zn_2SiO_4) from Aqueous Solution: Adsorption of hexanethiol on willemite from solution provides some very interesting results. Fig. 6 is the differential spectrum of the surface of willemite containing the adsorbate. The striking difference between this spectrum and those obtained for ZnO and ZnS is the fact that the —OH bands at 2.9 and 6.2μ are not inverted. The spectrum, which was reproduced many times, invariably indicates an increase in concentration of surface —OH groups. The —SH band at 3.95μ is missing, similar to the results obtained for ZnS and ZnO. Also very definite over-compensation bands appear at 9.4 and 12.2μ. The dotted spectrum of Fig. 6 is that of willemite alone. The wide band between 8.5 and 12.5μ is caused by a series of Si—O vibrational bands. The inverted bands at 9.4 and 12.2μ indicate that the surface of the solid itself is disturbed by the adsorbate, thus affecting some of the Si—O vibrational bands. The character of this spectrum suggests that the adsorbate breaks surface bonds upon

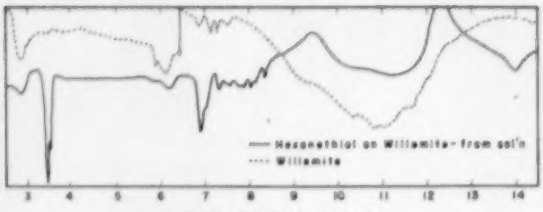


Fig. 6—Infrared spectrum of n-hexanethiol adsorbed on willemite from aqueous solution.

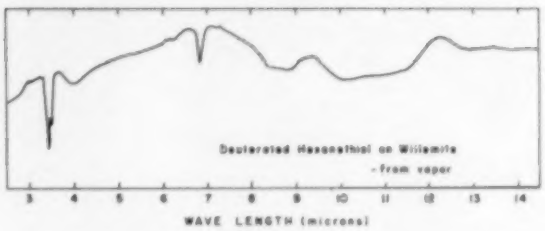
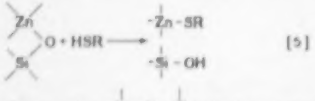


Fig. 8—Infrared spectrum of deuterated n-hexanethiol adsorbed on willemite from vapor phase.

adsorption and in so doing creates surface OH groups with an associated variation in the SiO vibrational bands. In the University of Utah laboratory such structural shifts as would occur by disturbing the Si—O vibrational frequencies have been detected for the adsorption of ammonia on several cracking catalysts and silica gel.[4]

Willemite is an orthosilicate in which both zinc and silicon are four-coordinated with the oxygen atoms in the lattice. Zinc and silicon tetrahedra share one corner only in the ortho-silicate structure. The observed spectrum may be explained in terms of an adsorption process, Eq. 5, that breaks the bonds linking these two tetrahedra together:



[5]

The shift from the $-\overset{|}{Zn}-O-\overset{|}{Si}-$ linkage to the Si—OH bonds explains the inverse peaks observed at 9.4 and 12.2 μ. Breaking of the oxygen bridge results in formation of the Si—OH band, causing the observed increase in surface hydroxyl groups. The adsorption process also destroys the S—H band through formation of the Si—OH groups, explaining the absence of the S—H stretching band (3.95μ) for adsorbed hexanethiol.

These results can be explained only by neutral molecule or perhaps ion-pair adsorption. Clearly a competitive reaction between the collector anion and hydroxyl ions such as is indicated in Eq. 3 cannot explain the spectrum obtained for solution phase adsorption of hexanethiol on willemite.

Adsorption of Deuterated Hexanethiol on Willemite from the Vapor Phase: A portion of hexanethiol was placed in contact with D_2O in a separatory flask for 48 hr. Virtually complete exchange between the hydrogen of the mercaptan and deuterium was achieved within this time. Fig. 7 is the spectrum of the deuterated mercaptan. The dotted spectrum was

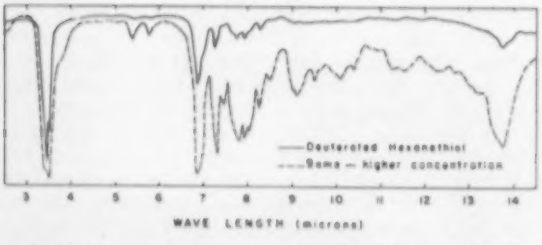


Fig. 7—Infrared spectrum of deuterated liquid n-hexanethiol. Solid curve 0.012 mm cell path length, dotted curve 0.05 mm path length.

that obtained using a cell path length of 0.05 mm compared to the spectrum with a 0.012 mm path length (solid line). As expected, the S—H stretching band is shifted approximately by $\sqrt{2}$ to 5.3μ. The S—H band at 3.95μ is missing, indicating almost complete exchange. The deuterated mercaptan should, according to Eq. 5, produce surface —OD bands. This band should grow up at approximately 4μ, since it also should be shifted by $\sqrt{2}$. The spectrum obtained for vapor phase adsorption of the deuterated hexanethiol on willemite is shown in Fig. 8. A definite peak, obtained at approximately 3.9 to 4.1μ, is broad and characteristic of that obtained for surface water bands. Again the structural Si—O shifts may be detected at 9.4 and 12.2μ. These results are identical to those expected according to Eq. 5, further substantiated by the formation of surface —OD bands.

Summary

Vapor phase and aqueous phase adsorption of n—hexanethiol on ZnO and ZnS as indicated by infrared spectroscopy involves a process resulting in a decrease in surface hydroxyl concentration. Invariably the mercaptan S—H bond is destroyed in the adsorption process. The vapor phase reaction undoubtedly involves neutral molecule adsorption. The adsorbed mercaptan reacts with the surface —OH group forming the zinc mercaptan salt, splitting out a molecule of water in the process. Identical results were obtained for aqueous phase adsorption of hexanethiol on ZnO and ZnS. The spectrum obtained and the very small dissociation constant of the mercaptan (1.9×10^{-11}) suggest neutral molecule adsorption. Similar results, however, would be expected for ion-pair adsorption or hydroxyl-mercaptan anion competitive adsorption. It appears from the results of this study that adsorption of collectors on mineral surfaces containing hydroxyl groups takes place by a process similar to esterification, in which the driving force is largely contained in formation of water.

Adsorption of hexanethiol on willemite (Zn_2SiO_4) breaks surface oxygen bridges forming $-\overset{|}{Si}-OH$ and $-\overset{|}{Zn}-SR$ surface groups. Confirmed by use of deuterated hexanethiol resulting in the formation of $-\overset{|}{Si}-OD$ groups at the surface, these results can be explained only by neutral molecule adsorption from the vapor phase. In the aqueous phase neutral molecule or ion-pair adsorption is consistent with the observed spectra. Hydroxyl ion competition is clearly incompatible with the infrared spectra obtained. It appears that surface reactions must involve processes maintaining strict electrical neutrality. This has long been recognized as a fundamental requirement for bulk reactions.

Application of infrared spectroscopy provides a powerful additional tool for the study of solids of large surface area. New techniques render this tool readily applicable to mineral systems such as are encountered commercially. Advantages gained are associated with the fact that the adsorbate may be examined *in situ*. In so doing it is possible to detect characteristic vibrational bands, gaining valuable information regarding the type of bonding present following adsorption. Differential spectroscopy, made possible by the pellet technique, provides a sensitive means for observing changes in both the adsorbate and the adsorbent which take place upon adsorption.

Acknowledgments

The authors wish to thank the Purex Corp. for financial support of this work. Thanks are also extended to Henry Eyring, dean of the graduate school, Carl J. Christensen, coordinator of research, and John R. Lewis, head, dept. of metallurgy, for their support and interest in this study.

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This study represents a portion of the work completed by E. M. Eyring in partial fulfillment of requirements for the Master of Science degree in chemistry at the University of Utah.

Discussion sent (2 copies) to AIME by Aug. 31, 1956, will be published in MINING ENGINEERING and in AIME *Transactions*, Vol. 205, 1956.

Human Response to Industrial Blasting Vibrations

by Jules E. Jenkins

IN the past quarter century the seismograph has played an increasingly important role in evaluating vibratory effects transmitted to adjacent communities by industrial blasting operations. In this period research has advanced through two major phases—from falling pin seismometers to three-component recording seismographs. The initial phase of this research is now regarded as the *era of recording*, during which the seismograph merely recorded occasional events. It has emerged from this stage into the *era of control and analysis* and in this later period has been recognized as a full-fledged industrial tool.

Although the falling pin machines of earlier years indicated probable amplitudes of industrial blasting vibrations, they gave no reliable measure of such displacements, nor did they provide essential information concerning direction or frequency of motion.

The first contribution of the three-component recording seismograph was the *weight-for-distance* formula developed by the U. S. Bureau of Mines. This empirical formula enabled users of commercial explosives to determine beforehand the probable vibratory effects of a blast when the weight of the explosives charge and the distance from the shot point were known. This formula has withstood the test of time and is still applicable for blasts detonated by instantaneous methods.

The second major contribution of the three-component recording seismograph followed the introduction in 1945 of short period or millisecond delay detonation. This might be described as a two-stage contribution, the first of which revealed a 50 pct reduction in the vibration magnitude when millisecond detonation was used rather than instantaneous discharge. This second part of this double contribution led to the discovery of the *vibration dip* phenomena, see Fig. 1.

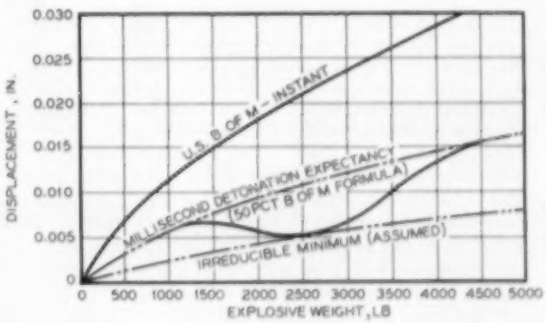

Fig. 1—Vibration dip phenomena. Vibrational effects conform to expectancy up to 1250 lb.

In Fig. 1 the horizontal coordinates indicate the displacements amplitude in inches, and the vertical coordinates, the explosive weight in pounds. The top curve is that of the U. S. Bureau of Mines for instantaneous detonation. Its curvature conforms to the two-thirds power of the explosive weight as provided in the *weight-for-distance* formula; hence the vibratory effects increase in proportion to an increase in the explosive charge. If the other curves shown below are temporarily disregarded, it may be considered that the entire area below this top curve represents both useful and waste energy.

As mentioned previously, when millisecond delay detonation was first investigated it was found that in general the U. S. Bureau of Mines curve could be decreased by about 50 pct, or to the position of the middle curve. At the outset it was assumed that this curve would likewise increase in some proportion to the explosive charge, as did the top curve, that is, the curve would indicate a continuous upswing. As seismographic data accumulated this assumption was found to be erroneous in part. It was discovered that

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TP 4214A. Manuscript, Feb. 15, 1955. Chicago Meeting, February 1955.

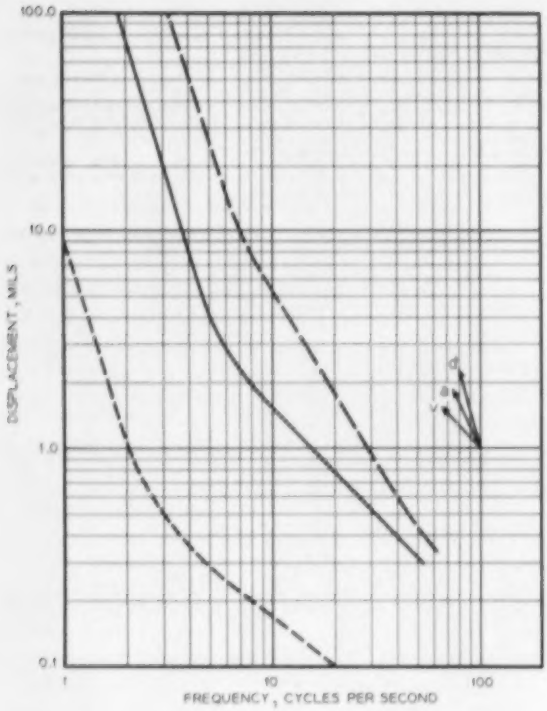

Fig. 2—Human response to vibration for vertical direction and standing position. Reading from lower left, curves represent *perceptible, objectionable,* and *uncomfortable* reactions of group of 15 persons. Top arrow at right indicates rate of change of acceleration; middle arrow, acceleration; and bottom arrow, velocity. (Based on Reiher and Meister.)

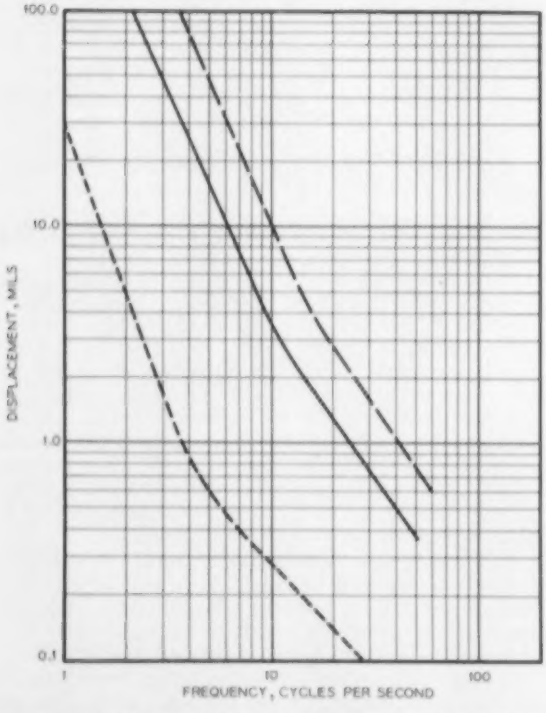

Fig. 3—Response for horizontal direction and standing position. From lower left, curves represent *perceptible, objectionable,* and *uncomfortable.* (Based on Reiher and Meister.)

for the smaller explosive loadings, vibrational effects conformed to expectancy only up to a point. In Fig. 1 this point is indicated at 1250 lb. Thereafter the vibrational effects assumed a downward trend that continued to the bottom of the dip, which is shown at 2500 lb. After reaching this low point, the curve rose again until it leveled somewhere near expectancy for the higher loadings. There is good reason to believe that this curve continues to the right and beyond the limits of the graph in a series of smaller dips. As of now loadings are up to 70,000 lb, but blasts of this size are few and far between. Consequently data is scant for loadings over 25,000 lb.

For the purpose of illustration, the dip is shown in Fig. 1 as extending over a 100 pct increase in the explosive loadings. In actual practice, the increase in the explosives loading varies from 20 to 80 pct, depending on individual blasting procedures and local terrain conditions.

When explanation is sought for this dip phenomena, the laws governing distribution of energy seem to apply best. Blasting vibrations transmitting beyond the shot point are classified as residual energy, which consists of two parts, irreducible minimum and waste. If the low point of the displacement dip is accepted as the point below which it is not possible to reduce, and if a final curve is drawn through this low point to the family of curves, the values falling on this curve represent the irreducible minimum—in other words, the amount of energy that must be driven into the earth to effect a new line of cleavage. It is with this waste energy that human response to industrial blasting vibrations begins.

In the course of these seismographic studies, investigators have been constantly alert to the wide discrepancies between reported vibration intensities based on human evaluations and actual intensity as determined by instrumental measurement. Although it was early recognized that these human factors extended into the fields of physiology and psychology, investigators refrained from straying into these fields, believing them outside their professional scope. Nevertheless, the point has been reached where someone must lead the way.

Investigations of human reaction to vibration are largely confined to steady state motion, for example, the riding qualities of vehicles and vibrations in ship structures and aircraft. At first reading, published reports are confusing and contradictory, but on further study each report is found to have merit within the limits investigated.

Unfortunately, aside from the fact that they provide valuable knowledge concerning human reaction to constant agitation, these reports say nothing of the transient or impact forces that occur as the result of an embedded dynamite blast. However, they do disclose interesting trends, which undoubtedly will be defined further in impact investigations.

The work of Reiher and Meister at the Institute of Technology in Stuttgart, Germany, in 1931 and 1935 provides important clues in this direction. These researchers subjected a group of 15 persons 25 to 40 years old, of various occupations, to the continuous oscillations of a vibrating platform, through amplitudes from four ten-thousandths of an inch to 4 in., at frequencies from 1 to 70 per sec. They exposed each subject to various vibrational modes for 8-min intervals, during which the subject stood or reclined. When the subject reclined, motion was applied both parallel and transverse to the axis of

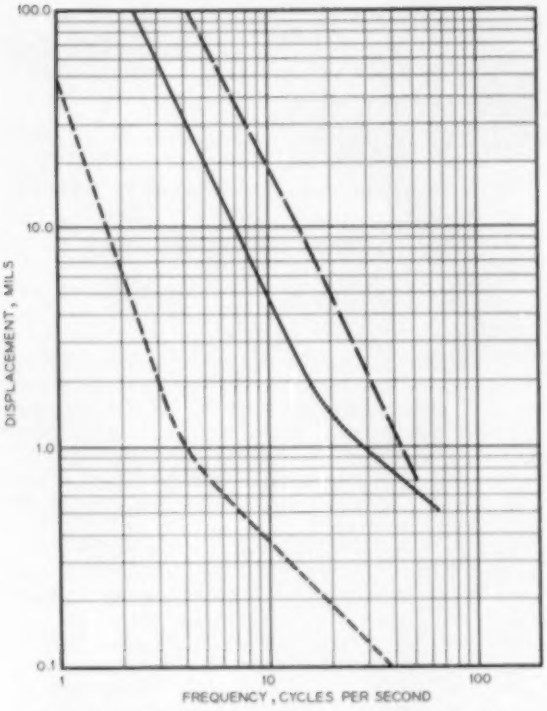

Fig. 4—Human response to vibration for vertical direction and reclining position. From lower left, *perceptible, objectionable,* and *uncomfortable.* (Based on Reiher and Meister.)

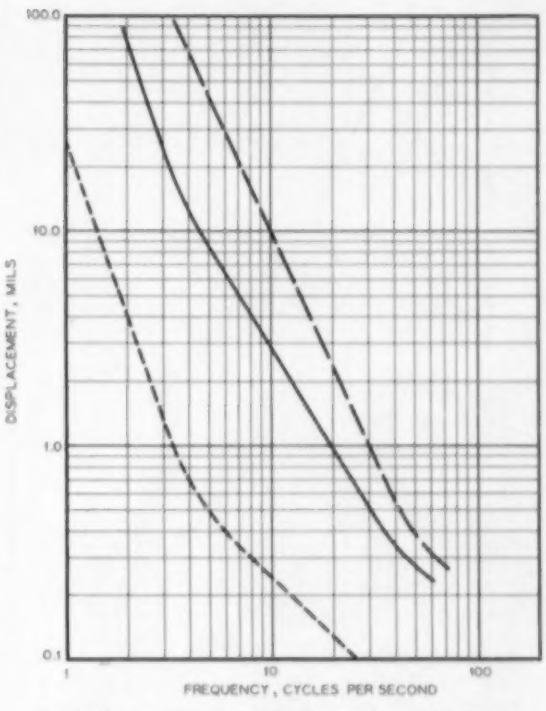

Fig. 5—Human response to vibration for longitudinal direction and reclining position. From lower left, *perceptible, objectionable,* and *uncomfortable.* (Reiher and Meister.)

the body. After each test the sensations experienced by the subject were rated as 1) not noticeable, 2) barely noticeable, 3) well noticeable, 4) strongly noticeable, 5) objectionable, and 6) uncomfortable. Thus between categories 1 and 3 the threshold of irritation was established, and between categories 4 and 6 the threshold of discomfort.

As was expected, the degree of sensation varied widely between subjects. Furthermore, the same sensation was not always experienced by the same subject reacting to the same vibrations. Even so, the test results were carefully plotted and a wealth of valuable data obtained. These data are condensed in Figs. 1 through 6, which reproduce only three of the six curves by Reiher and Meister, namely, the second, fourth, and sixth, indicated as *perceptible, objectionable,* and *uncomfortable.* Horizontal lines in Fig. 2 and in the illustrations to follow indicate vibration amplitudes in mils, or thousandths of an inch, and vertical lines the frequency in cycles per sec. The bottom curve represents *perceptible,* the middle curve *objectionable,* the top curve *uncomfortable.* The arrows on the right represent three vibrational characteristics. The top arrow indicates the rate of change of acceleration, sometimes referred to as *jerk,* the middle arrow indicates acceleration, and the bottom arrow, velocity. When the slope of the curve corresponds to the direction of one of these arrows, it indicates that the human sensations within this segment of the curve are stimulated and reacting to the vibration characteristic shown by the arrow. This reveals that in the lower frequencies, below 5 cycles, the reaction tends to conform to the rate of change in acceleration; in the middle frequencies, 5 to 10 cycles, to acceleration; and in the higher frequencies, above 10 cycles, to velocity.

In this case, at 10 cycles (bottom-center reading up) the effect of vertical movement applied to a person standing must be increased nine times over *perceptible* before becoming *objectionable,* and 32 times before becoming *uncomfortable.* The difference between objectionable and uncomfortable represents an increase of three and a half times.

When the person remains standing while being subjected to horizontal movement, Fig. 3, the motion must be increased 13 times over *perceptible* to become *objectionable,* and 39 times before becoming *uncomfortable.* The difference between the *objectionable-uncomfortable* limits is three times, or approximately the same differential as that for vertical movement. In general, when someone is standing, twice as much horizontal as vertical movement is required to invade *objectionable-uncomfortable* limits.

When the subject reclines and is moved in the vertical direction, Fig. 4, an increase of 13 times over *perceptible* is required to become *objectionable,* and an increase of 51 times to become *uncomfortable.* When this reclining position is compared with the standing position, it is found that three times as much vertical movement is required to invade the *objectionable-uncomfortable* limits. In other words, the body is more sensitive to vertical vibration when in an upright position. When the subject is still reclining and being moved horizontally, parallel to the axis of the body, Fig. 5, the reaction is similar to that for horizontal-standing.

When the subject reclines and motion is applied transverse to the axis of the body, human reaction is more pronounced than for any of the various combinations with the exception of vertical-standing.

In summary, the human body under steady state agitation is most sensitive to vertical motion when

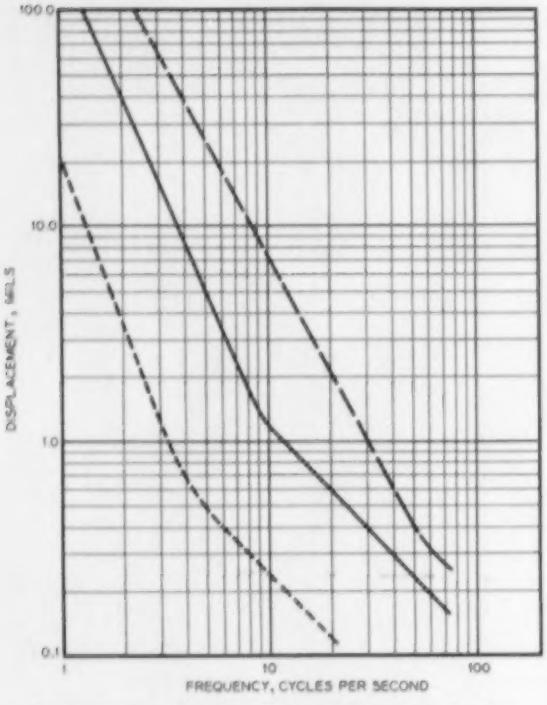

Fig. 6—Human response to vibration for transverse direction and reclining position. From lower left, *perceptible, objectionable, and uncomfortable.* (Based on Reiher and Meister.)

the person is standing and least sensitive when he is reclining, as shown in the left diagram of Fig. 7. When the movement is confined to the horizontal, the reclining person is most sensitive to motion applied transverse to the body axis, as shown in the lower diagram. Sensations of this kind are experienced in sleeping cars of trains.

Finally, although this discussion is confined to amplitudes having a frequency of 10 cycles per sec, the importance of frequency on these human reactions must not be overlooked. For example, in Fig. 6, if the frequency were increased six times to 60 cycles, the *objectionable* sensations would be experienced at 0.19 mils, or about two ten-thousandths of an inch. This motion is but one sixth of that considered *objectionable* at 10 cycles. Furthermore, *objectionable* motion at 60 cycles would not be *perceptible* at 10 cycles. Now, if frequency is decreased to 5 cycles, the *objectionable* sensation would not be experienced until an amplitude of 5 mils was reached, or four times the movement considered *objectionable* at 10 cycles.

If these curves are extended to the upper edge and beyond the limits of the graph, into the range of earthquake frequencies, amplitudes of the order of inches would be required to produce the same degree of human sensation experienced by amplitudes of less than one thousandth of an inch in the blast frequency range. This offers some explanation of the claim frequently advanced that the results of

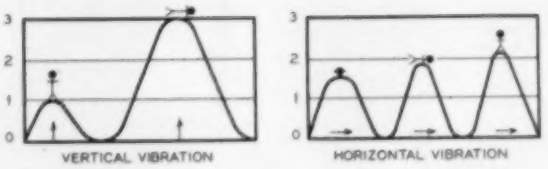

Fig. 7—Objectionable level. Human response at 10 cycles per sec.

some industrial blasts are worse than an earthquake.

One important phase of human reactions, not covered by the investigations of Reiher and Meister, has to do with what is called the *startle reaction.* This reaction, induced by the ear, is known to bring about momentary increases in heart rate and respiration. The source need not be loud or intense, but only unexpected. It has been well established that these sudden disturbances exert profound influence on a person's decision that he detests a certain sound, and incidentally, those responsible for making it.

Not only the ear is involved. Skin areas, muscles, tendons, joints, and internal organs are excited as well. The individual or collective effects on these parts of the body frequently cause people to conclude that the earth or building supporting them is shaking violently, when, as a matter of fact, they are actually being moved through microscopic dimensions.

It must be remembered that the average person can feel vibrations that are *one hundredth* to *one thousandth* parts of the magnitude necessary to damage structures. Until these facts are firmly implanted in the public mind, those whose business requires them to discharge blasts will undoubtedly be confronted, from time to time, with claims for structural damage, which are in fact based on misconceptions arising from human reactions.

Although these facts point to the road ahead, they do not in themselves hold any immediate application to industrial blasting procedures. They do, however, clearly disclose the urgent need for further investigations into the complexities of human response to impact. There is every reason to believe that the body can withstand a far greater degree of shock under impact than it can under prolonged agitation.

The Vibration Measurement Engineers organization has made an exhaustive review of its files and endeavored to correlate some of the measured seismographic data with statements made by people the company regards as chronic complainers. On the basis of this approach, there is reason to believe that industrial blasting vibrations might be increased as much as five to ten times that of steady state motion before the *objectionable-uncomfortable* zones are invaded.

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Discussion of this paper sent (2 copies) to AIME by Aug. 31, 1956, will appear in MINING ENGINEERING and in AIME *Transactions,* Vol. 205, 1956.

Correction

In the January 1956 issue: TP 4137B. Adsorption of Ethyl Xanthate on Pyrite. By A. M. Gaudin, P. L. de Bruyn, and Olav Mellgren. P. 65. Since P. L. de Bruyn is now a member of the AIME, the word "associate" should be removed in describing his status. The address of the third author, Olav Mellgren, should read: 74-2 Chingola, N. Rhodesia. P. 68, second paragraph, last line: the word "300-fold" should be replaced by "3000-fold."

Also in the January 1956 issue: TP 4158B. Energy Transfer by Impact. By R. J. Charles and P. L. de Bruyn. The authors wish to express their disapproval of the omission of the abstract from the article, since they believe that an abstract fulfills a very important function in every paper.

Precipitation of Metal from Salt Solution By Reduction with Hydrogen

Early work on chemical precipitation of metals from metal salt solutions is reviewed. The chemistry and thermodynamics of precipitating copper, nickel, cobalt, and cadmium metals by reaction with hydrogen are discussed. Mechanisms of metal precipitation, nucleation, growth, and agglomeration are reviewed, as well as some solubility phenomena of gases and solids at elevated temperatures. Experimental data presented deal mainly with the reaction of copper sulfate solution with H_2.

by F. A. Schaufelberger

METAL can be recovered from a leach solution either indirectly by precipitation as a compound that is later reduced or directly by electrolysis, cementation, or chemical reduction, for example, with hydrogen. The widely used electrowinning processes suffer from the inefficiency of converting fuel to low voltage dc and from low current efficiency when complete recovery is attempted. With the batch hydrogen reduction methods outlined here, a unit of fuel converted to hydrogen in modern gas reform plants will make two to three times as much metal as can be obtained with electrolysis. With continuous hydrogen reduction, four to six times as much metal is obtained.

Precipitation by reducing a metal salt solution with H_2 has been known for almost 100 years.[1] Commercial use of the process, however, awaited the research and development program initiated by Chemical Construction Corp. Within the last few years this program has brought about construction of commercial plants, listed on page 545.

Ideal hydrogen reduction will precipitate a pure metal from a solution obtained by commercial leaching methods at a rapid rate without excessive temperatures and pressures. The metal precipitate will be of desired size and density, less than a percent left in the vessel being deposited on the wetted parts.

The present discussion will outline Chemical Construction Corp.'s early development program and will discuss the chemistry and mechanics of reducing copper, nickel, cobalt, and cadmium from solution by H_2. Work on selective reduction of nickel from cobalt has been described earlier.[2]

The article is not concerned with precipitation of copper by disproportionation of cuprous solutions where yield is limited[3] to 50 pct.* It does not discuss use of gaseous reducing agents other than hydrogen, such as SO_2,[4] which may lead to contamination with sulfur, or CO,[5] which is considerably more expensive than hydrogen and produces a gaseous reaction product, CO_2.† The article does not include use of other nongaseous reducing agents,[7] which are far more expensive than CO. Also, notes on the history, engineering design, and performance of commercial plants have been given elsewhere.[8]

It will be shown that a pure metal can be precipitated by reduction with hydrogen from solutions obtained by commercial leaching methods when the solution composition is controlled and proper acidity, proper metal ion concentration through controlled complex formation and hydrolysis, and adequate agitation for suspension of metal and for transfer of reducing gas are maintained. Reasonable temperatures and pressures are used which are less excessive than those used by previous workers, although they are maintained at above equilibrium values because of process kinetics.[9]

It has also been found necessary to induce nucleation and to control growth and agglomeration in order to get a powder product of suitable physical properties.

Review of the Literature

Muller, Schlecht, and Schubardt of I. G. Farben[10] claimed a successive reduction of silver, copper, nickel, cobalt, and zinc from ammoniacal solution by applying progressively higher temperatures and hydrogen pressures. Metals produced by this technique were not pure, since no attempt was made to adjust the solution composition between the different reductions. It is doubtful that the zinc product contained any metallic zinc.

The major contribution to the literature[11] was made by the Ipatiews,[12] who prepared platinum, iridium, copper, nickel, cobalt, lead, tin, arsenic, antimony, and bismuth at often extreme conditions, up to the critical temperature of water (373°C), at 1500 to 7500 psi, for as long as several days. Even cadmium and zinc were claimed to have been observed in trace quantities precipitated together with basic salts. In the early experiments solutions were not stirred. Extensive formation of oxides and basic salts occurred, probably long before sufficient hydrogen could be supplied in the unstirred liquid to decrease the metal concentration in solution by reduction. The importance of hydrogen pressure was

* Copper precipitation by disproportion, $2\,Cu^{+} = Cu^{\circ} + Cu^{++}$, is due to instability of the cuprous ion. Cuprous ion can be stabilized by formation of complexes, e.g., with CO or ammonia. Removal of CO from a $Cu_2(CO)_2SO_4$ solution by boiling or by application of vacuum, or the neutralization of a cuprous ammine solution with acid, will cause copper precipitation.

† Formation of toxic nickel tetracarbonyl by reaction of CO on ammoniacal nickel solutions (180°C, 300 psi CO) is discussed in Ref. 6.

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TP 4239BD. Manuscript, Nov. 22, 1954. Chicago Meeting, February 1956.

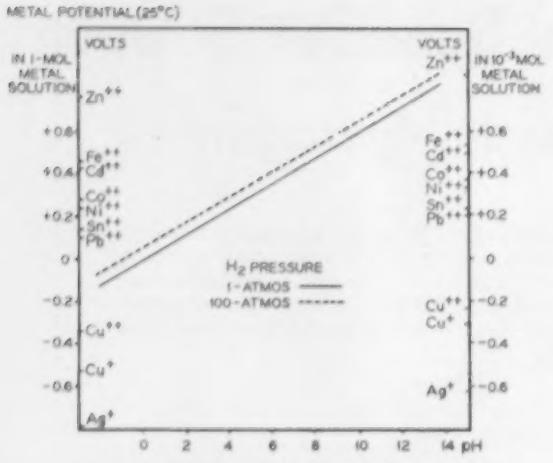

Fig. 1—Potential of 1 and 10^{-3} molar metal solutions, and hydrogen potential at varying pH (25°C).

over-emphasized. The pioneer work of the Ipatiews has been continued in Russia[16] with the most recent report by Tronev, Bondin, and Khrenova[14] on reduction of copper, nickel, cobaltous and cobaltic ammoniacal chloride and sulfate solutions with hydrogen. Most of this work, however, was carried out in unbuffered solutions.

Reduction Equilibrium

A. Calculation of Equilibrium: Reduction of divalent metal ions such as copper, nickel, cobalt, lead, and cadmium by hydrogen may be written as

$$M^{++} + H_2 \rightleftarrows M^{\circ} + 2H^{+} \qquad [1]$$

with an equilibrium constant K (neglecting activity coefficients)

$$K = \frac{[H^{+}]^2}{[M^{++}]P_{H_2}} \qquad [2]$$

where brackets denote molar concentrations and P the pressure in atmospheres.

The degree to which this reaction occurs is given by the familiar thermodynamic equation for the free energy change ΔF (cal.) or more conveniently, the potential E (volt):

$$\Delta F = -23{,}060\ n\ E = -RT \ln K. \qquad [3]$$

Reductions of copper, nickel, and cobalt solutions with H_2 attain commercially attractive rates only at temperatures higher than 70°C. Extrapolation of thermodynamic data is certain to be speculative, but actual data at elevated temperatures are not available. It is therefore convenient first to calculate E for room temperature conditions, then to attempt to approximate E for elevated temperatures, and finally to discuss briefly the effect of variation in experimental conditions.

Fig. 1 plots the variation in E at 25°C for 1 and 0.001 M concentrations of the metal ions of interest, and also the hydrogen potential at various pH values and at 1 and 100 atm H_2 partial pressure (fugacity at 100 atm and 25°C = 106.1 atm[15]). Of course, the metal potential is affected by pH only indirectly, since the metal concentration is a function of pH due to complex or hydroxide formation. For reduction to proceed, the hydrogen potential must exceed the metal potential. It appears possible to reduce metals up to cadmium, perhaps including iron but

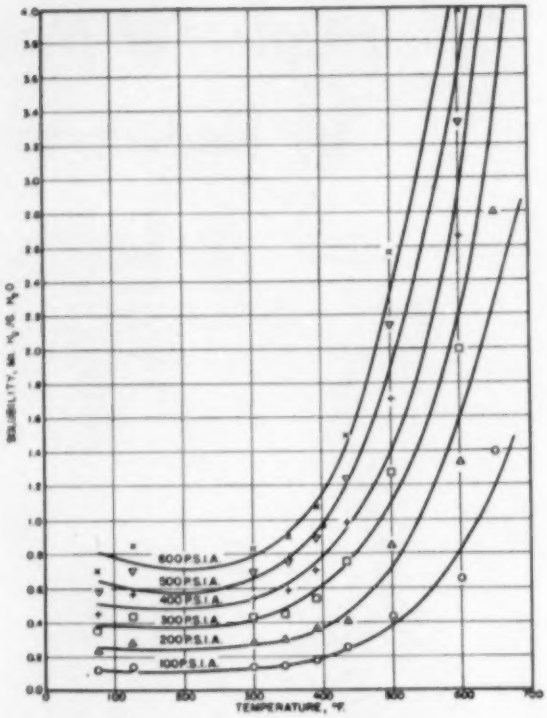

Fig. 2—Solubility of hydrogen (standard temperature and pressure) in water at varying temperatures and pressures. (From *Industrial and Engineering Chemistry*, 1952, vol. 44, p. 1147.)

hardly zinc. Cadmium turned out to be the least noble metal to be precipitated in pure form at industrially satisfactory yields and rates.

It will be noted from Fig. 1 that a hundredfold increase in hydrogen pressure is comparable to an increase of one pH unit.

Calculating reduction equilibria in aqueous solution for elevated temperatures is difficult in view of the uncertainty in estimating the effect of temperature on the entropies of solutes, ions, or undissociated molecules. Since there are ions on both sides of the equation, temperature changes in entropies were assumed roughly to cancel out. This is admittedly not a very good approximation, but it appears as good as any in the absence of experimental data. Eq. 4 was used to calculate reduction equilibria using room temperature data:

$$\Delta F_T = \Delta F_{298°K} - \Delta S_{298°K}\ (T - 298) \qquad [4]$$

It is based on the assumption that the reaction entropy at temperature, T, ΔS_T, is equal to the reaction entropy at 25°C, $\Delta S_{298°K}$. Most of the data were taken from Latimer,[18] but for cobalt and nickel the data obtained by Haring[17] were selected. Data by Cobble[19] and Yatsmirsky-Grafova[20] were used to compute entropies of the ammine complexes.

B. Solubility of Solids and Gases in Water at High Temperatures: Solubility of most metal compounds in water increases with temperature up to 120° to 150°C but then decreases with further increasing temperature and becomes zero at temperatures approaching the critical temperature of water, 373°C.[25] Even at 250°C, solubility of most metal sulfates has decreased to less than 1 gpl. At elevated temperatures solubility of basic metal salts and hydroxides is also markedly decreased, and they

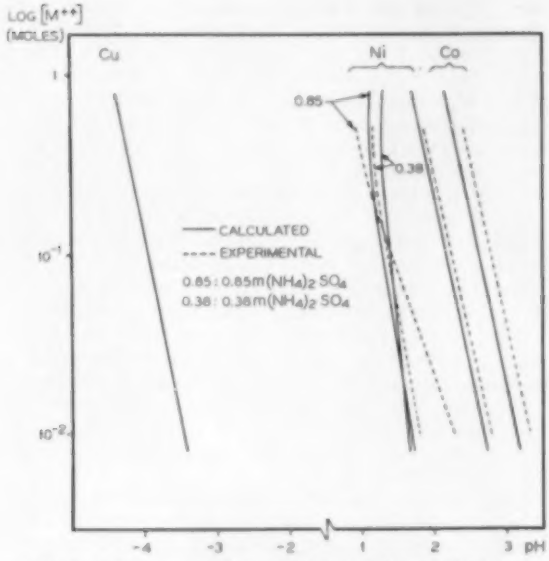

Fig. 3—Reduction equilibria of $CuSO_4$, $NiSO_4$, and $CoSO_4$ in aqueous solution with H_2 (490 psig partial pressure). Reductions carried out at 200°C. pH measurements taken after quenching to 25°C. Calculated figures are computed for 200°C for solutions containing no $(NH_4)_2SO_4$. For Ni solutions containing $(NH_4)_2SO_4$, computation includes cooling to 25°C. (Reprinted from *Transactions of the Institution of Mining and Metallurgy*, London, 1954/1955, vol. 64, p. 379.)

will often precipitate out at reduction temperatures from solutions of considerably higher acid strength than at room temperature. The experience of early investigators is so full of references to such precipitates that it is easy to see the importance of controlling solution composition and using temperatures below 250°C to achieve pure metal precipitates. Of course, some high temperature precipitates may reduce as a solid phase and others may redissolve as the reduction proceeds; others may influence the reduction favorably, as hydrogen activators or as nuclei. In reduction from ammoniacal solutions hydrolysis is particularly important, but the present discussion will have to be limited to the reduction of the metal ion only.

Gas solubilities in water, on the other hand, decrease with temperature increase up to about 80°C and then markedly increase with further increasing temperature,[28] see Fig. 2.

The unusual trend in gas solubility has been associated with the change in the structure of water from quasi-crystalline at room temperature to close-packed at higher temperature. At 200°C the structure of water is apparently similar to that of organic solvents, such as benzene, at room temperature[29] and shows similar solvent characteristics. The structure of water is noticeably affected by pressures only above 5000 psi, or above reduction pressures under consideration here.[30]

C. Effect of Reaction Variables: Inspection of Eq. 2 indicates that thermodynamics of metal reduction is favored by a low H^+ concentration, a high M^{++} concentration, and high hydrogen pressure.

1. *Illustrative Experiments:* Table I outlines the results of various experiments qualitatively, illustrating the effect of M^{++} and H^+ ion concentrations on reduction of nickel by hydrogen in various environments. A low nickel concentration was chosen to prevent complication by precipitation of basic nickel salts. The technique is described on page 548.

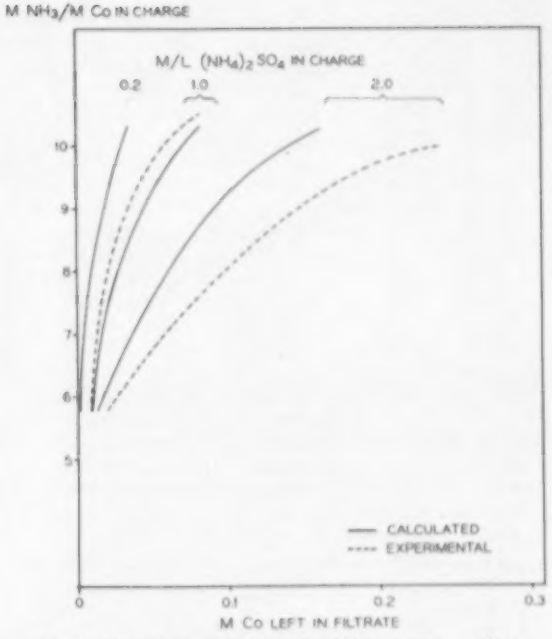

Fig. 4—Reduction equilibrium of cobaltous ammine sulfate with H_2 at 200°C and 500 psig H_2 partial pressure.

With nickel chloride, only 5 pct of the nickel was reduced because the H^+ ions that formed during the reduction—HCl being completely dissociated—had reduced the pH to a point where nickel reduction had to stop. In the experiment with nickel sulfate, the moderately stable bisulfate ion is formed and about 10 pct of the nickel was reduced before the solution became too acid. When ammonium sulfate was added to the nickel sulfate, the hydrogen ions produced were increasingly neutralized by formation of bisulfate, and this buffering allowed half the nickel to be reduced before the solution was too acid.

When ammonium acetate was added, the hydrogen ions produced were tied up as largely undissociated acetic acid, and the solution was completely stripped of nickel without becoming too acid. Nickel sulfate plus pyridine reacts to form a weak nickel complex, but pyridine is not a strong enough base, so the solution became too acidic and the reaction stopped.

In the next experiment the nickel was complexed with a stronger base, ammonia, and the solution was buffered with ammonium sulfate. The ammonia raised the pH so that hydrogen became a stronger reducing agent, but it also decreased the concentration of nickel ion in solution by complex formation so that it was harder to reduce. This buffering and complexing also prevented nickel compounds from precipitating in the alkaline solution.

In the experiment with KCN, the concentration of

Table I. Hydrogen Reduction of Various Nickel Solutions*

Solution	Approximate Reduction at Equilibrium, Pct	Final pH, Room Temperature
$NiCl_2$	5	1.9
$NiSO_4$	10	2.2
$NiSO_4$-$(NH_4)_2SO_4$	50	1.6
$NiSO_4$-NH_4 acetate	100	4.1
$NiSO_4$-pyridine	70	2.0
$NiSO_4$-NH_4-$(NH_4)_2SO_4$	96	9.8
$NiSO_4$-KCN	0	11.0

* Ni = 0.2 *M*; temperature = 200°C; H_2 pressure = 1000 psig.

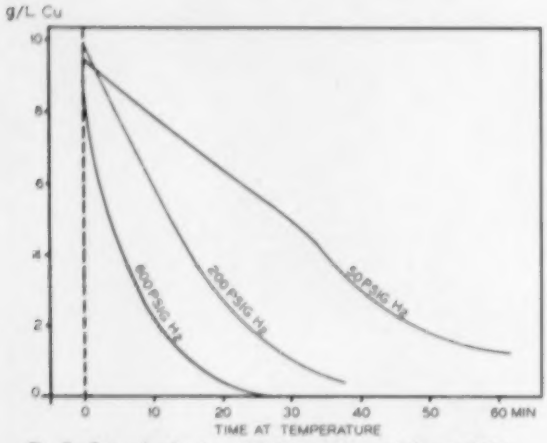

Fig. 5—Rate of reduction of $CuSO_4$ solution with H_2 at 204°C and with varying H_2 partial pressures. Charge, gpl: 10 Cu as sulfate, 5 Fe as sulfate, 120 H_2SO_4, 200 $(NH_4)_2SO_4$, and 80 Cu powder.

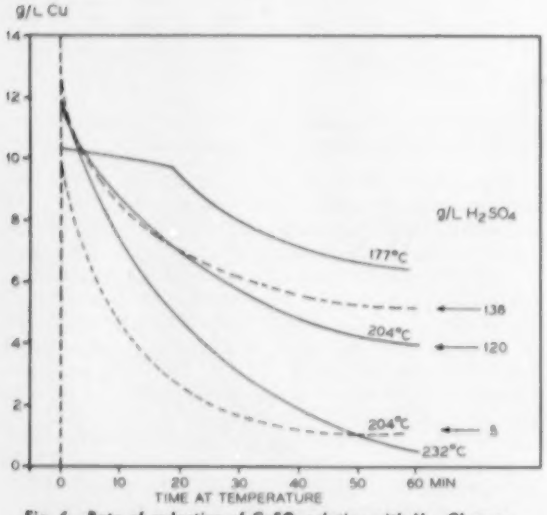

Fig. 6—Rate of reduction of $CuSO_4$ solution with H_2. Charge, gpl: 10 Cu as sulfate, 5 Fe as sulfate, 120 H_2SO_4, or as indicated, 80 Cu as powder; 200 psig. H_2 partial pressure. Varying temperatures (177°, 204°, 232°C at 120 H_2SO_4) and varying initial acid concentrations (5, 120, 138 gpl H_2SO_4 at 204°C).

nickel ion in solution was so low, because of CN^- complexing, that reduction could not be obtained even at pH 11.

2. *Effect of Hydrogen Ion Concentration:* The equilibrium concentrations of copper, nickel and cobalt sulfate were calculated from Eq. 4 and available room temperature data. Accordingly the free energies of reduction for $CuSO_4$, $NiSO_4$, and $CoSO_4$ are somewhat more negative, i.e., more favorable at 200°C compared with 25°C. The difference appears negligible for copper and amounts to not more than 1 to 2 kcal for nickel and cobalt.

Autoclave experiments were run to determine at 200°C and 490 psi hydrogen partial pressure the nickel and cobalt concentrations in equilibrium with hydrogen at various pH values.[3] Results of experiments and calculations are shown in Fig. 3. Ordinate values represent the molar concentrations of the dissolved metal. The calculated pH values pertain to the reaction temperature; the experimental pH values were of necessity obtained at room temperature after cooling a sample from the autoclave. However, the pH at room temperature is expected to approximate the actual hydrogen ion activity at 200°C because the stable HSO_4^- ion appears to become even more stable with rise in temperature. It will be noted that copper can be essentially quantitatively reduced in strong acid solutions with a calculated pH of −3, or 1000 *M* acid. Although this could not be checked experimentally, complete reduction has been obtained from sulfuric acid solutions as strong as 20 pct. The data also show that cobalt and nickel can be reduced to 0.01 *M* (0.6 gpl) at pH values of 3.3 and 2.75 respectively.

Selective reduction at controlled acid strength is indicated for commercially acceptable separation of copper, nickel, and cobalt from each other and from other metals likely to be found in leach liquors, for example, iron, zinc, aluminum, and magnesium. Experiments in stepwise selective reduction which show the sharp separation of two metals with potentials as close as those of nickel and cobalt have been reported in another paper.[3]

Data on the effect of ammonium sulfate on reduction of $NiSO_4$ are also included in Fig. 3. With ammonium sulfate present, reduction appears to proceed to a lower pH value before equilibrium is reached. The pH is chiefly determined by the bisulfate-sulfate buffer system. The second dissociation constant of H_2SO_4, which is 1.25×10^{-2} at 25°C has been estimated to be 5.1×10^{-4} at 200°C. The hydrogen ion activity at reduction equilibrium therefore will increase upon cooling and furnish a pH somewhat more acid than the actual acidity at reduction conditions, satisfying the expression:

$$k_{200°C} = 1.25 \times 10^{-2} = \frac{[H^+]\,[SO_4^=]}{[HSO_4^-]}. \qquad [5]$$

In contrast to the above discussed reductions without ammonium sulfate, these equilibria with ammonium sulfate were calculated for room temperature, using Eq. 5 to convert the equilibrium computed for 200°C to room temperature.

For calculation purposes the activity of the metal ion at 200°C was assumed to be 1 and about 0.1 in the presence of ammonium sulfate, taking into account the formation of sulfate complexes.[4] Experimental data checked the calculated data closely enough to support the assumptions made.

Additional experiments studying the reverse reaction, the dissolution of metal in H_2SO_4 under H_2 pressure, gave an order of magnitude agreement. It is interesting to note that while nickel metal dissolves easily in 1 pct H_2SO_4 to produce H_2 gas at atmospheric pressure, 500 psi H_2 partial pressure will stop nickel dissolution in 1 pct H_2SO_4—0.5 *M* nickel solution.

3. *Effect of Metal Ion Concentration:* Metal ion concentration depends upon complexing and also pH which controls the precipitation of basic compounds.

Ammonia is an economical reagent for raising the pH of the solution and thus obtaining a more favorable thermodynamics for metal reduction. On the other hand, ammonia is also an efficient complexing agent for copper, nickel, cobalt, and cadmium, and the metal ion concentration in such solutions is therefore considerably diminished according to:

$$k_s = \frac{[M^{++}]\,[NH_3]^x}{[M(NH_3)_x^{++}]}. \qquad [6]$$

In Table II overall dissociation constants k_s and standard potentials are given for a number of metal

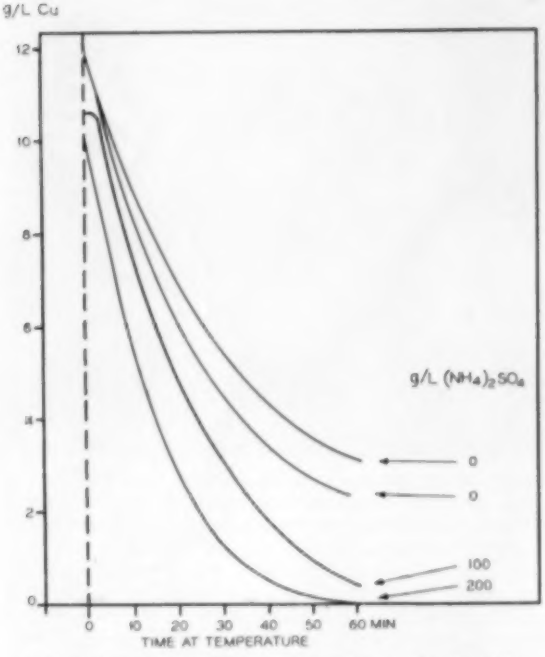

Fig. 7—Rate of reduction of $CuSO_4$ solution with H_2 (200 psig partial pressure) at 204°C, with varying sulfate concentration. Charge, gpl: 10 Cu as sulfate, 120 H_2SO_4, 80 Cu powder; 5 Fe as sulfate, except second curve from top: 15. $(NH_4)SO_4$ as indicated.

ammines. "$\frac{N}{Z}$ log k + log 55" data are also included. This expression was introduced by Bjerrum a few years ago[25] as a more convenient way for expressing the strength with which the ligands are held. N is the number of uniformly bound ligands, Z the corresponding coordination number, and k the mean complexity constant of the system.

It should be noted that due to the decrease of metal ion concentration the potential for the ammine is higher. However, the increase is less than the increase in the hydrogen potential with a change from acid to ammoniacal solution, and all these complexed solutions are thermodynamically reducible by hydrogen from ammoniacal solution.

In contrast to its behavior in acidic solution, bivalent cobalt in ammoniacal solution is more noble than nickel. This is due to the increased dissociation of the cobaltous ammine complex, the dissociation constant being 1.25×10^{-5}, as against 1.8×10^{-9} for nickelous ammine.

Also it appears that a certain nickel-cobalt separation can be obtained in acid but that in ammoniacal solution cobalt should reduce ahead of nickel. However, some selective nickel reduction has also been obtained in ammoniacal solution.[26, 27] This indicates that nickel reduces at a faster rate than cobalt.

The reduction of divalent metals from ammoniacal solutions may be written:

$$M^{++} + 2NH_3 + H_2 = M^{\circ} + 2NH_4^{+} \qquad [7]$$

or

$$K = \frac{[NH_4^{+}]^2}{[M^{++}]\,[NH_3]^2\,P_{H_2}} = \frac{[NH_4^{+}]^2 \cdot [NH_3]^{z-4}}{k_z \cdot [M(NH_3)_z^{++}]\,P_{H_2}}. \qquad [8]$$

Evaluating this equation at 200°C with room temperature data again requires the simplifying assumptions used with Eq. 2. This gives $K_{298°C} = 4.78 \times 10^{6}$ for the reduction of $Co(NH_3)_6^{++}$. Assuming $k_{298°C} = 4 \times 10^{-5}$, activity coefficient of $NH_3 = 0.5$ and of NH_4^{+} and the ammine complex = 0.2, the curves of Fig. 4 were obtained. These curves were checked by experimental reduction for 0.2 and 2.0 mols per liter $(NH_4)_2SO_4$.

The presence of k_z in the denominator in Eq. 8 indicates that reduction of weak complexes is favored. It is apparent that high concentrations of ammonium salts and free NH_3 (except in diammine reduction) adversely affect the reduction equilibrium, see Fig. 4.

Eq. 8 is valid only for one particular complex, whereas actually a number of different complexes exist that should be taken into account in an exact treatment of the system. It should be remembered that during the course of the reduction the total metal ion concentration decreases and $2M$ NH_3:M metal are converted to $(NH_4)_2SO_4$. For example, if a mol ratio of 4 NH_3:Co is used at the start, after 90 pct reduction the mol ration is 22 NH_3:Co. With increasing NH_3:Co ratio the complexing increases and cobalt ion concentration decreases much faster

Table II. Copper, Nickel, and Cobalt Ammines*

Standard Potential (25°C) (volts)			Dissociation Constant of Ammine	
Ionic Solution	Ammine Solution		k_z	$\frac{N}{Z}$ log k + log 55
	$Cu(NH_3)_4^{++}$	+ 0.12	1.38×10^{-13}	7.2
Cu^{++} — 0.337	$Cu(NH_3)_4^{++}$	+ 0.03	4.7×10^{-13}	8.0
Ni^{++} + 0.231	$Ni(NH_3)_6^{++}$	+ 0.49	1.8×10^{-9}	3.3
Co^{++} + 0.278	$Co(NH_3)_6^{++}$	+ 0.42	1.25×10^{-5}	2.6
	$Co(NH_3)_6^{+++}$		2.2×10^{-36}	7.6

* According to Latimer, Ref. 16.

than the total cobalt concentration. It would seem that the use of more than 2 NH_3:Co would be undesirable because the reduction would not go to completion. However, the lower ammines are not very stable at reduction temperatures, and an ammonia ratio initially somewhat above 2 may be required to prevent hydrolysis. Hydrolysis can also be minimized by adding $(NH_4)_2SO_4$. In commercial

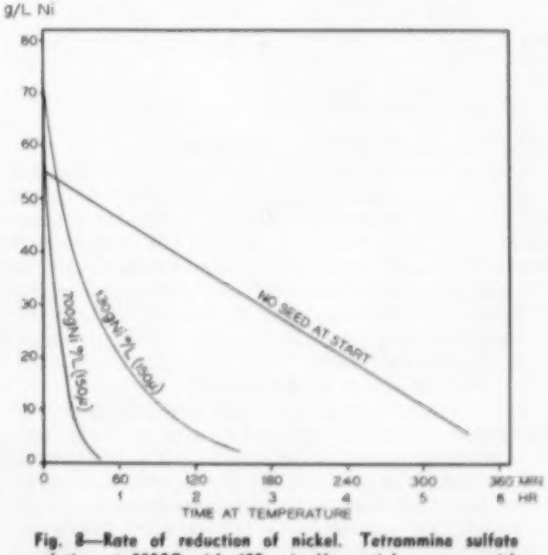

Fig. 8—Rate of reduction of nickel. Tetrammine sulfate solution at 232°C with 450 psig H_2 partial pressure, with and without nickel seed.

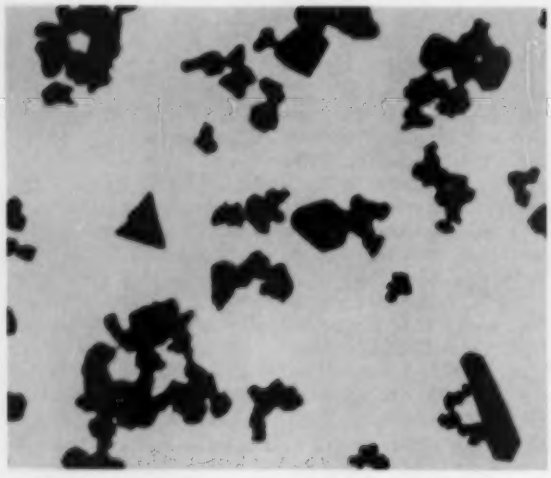

Fig. 9—Crystalline copper precipitated at 80°C from $Cu_x(CO)_ySO_4$ solution by slow disproportionation. X300.

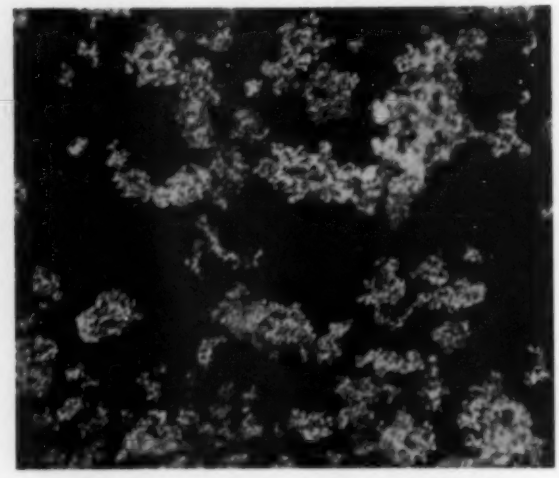

Fig. 10—Copper powder. H_2 reduced from $CuSO_4$-H_2SO_4 solution at 200°C. No seed. Agglomerates of crystals are result of precipitation by H_2 at high reaction rate in commercial-type reaction. X25.

practice, in cobalt reduction 3 NH_3:Co is used with 0.4 *M* $(NH_4)_2SO_4$ while in nickel reduction about 2 NH_3:Ni is used with up to 2 *M* $(NH_4)_2SO_4$. In other words, extra ammonia has been found better for cobalt and extra $(NH_4)_2SO_4$ has been found better for nickel in order to control both precipitation of nonmetallic compounds at the beginning of the reduction and the adverse effect of high NH_3:metal and high $(NH_4)_2SO_4$ at the end. The situation is even more involved when ammoniacal copper carbonate solutions (scrap copper leach liquor) are reduced. Copper in ammoniacal solution is present in two valence states and numerous complexes between Cu^+, Cu^{++}, NH_3, and carbonate exist. When CO is used as reducing agent, additional complexing of Cu^+ with CO occurs. Not only is the picture complex at the start; the situation becomes more complicated during the course of reduction as NH_3 and CO_2 are liberated, resulting in an increase of their partial pressures and a corresponding decrease of the partial pressure of the reducing gas.

4. *Effect of Hydrogen Pressure:* Data in Fig. 1 show that increasing hydrogen partial pressure has small effect on reduction equilibrium compared to the effect of pH. Use of high H_2 partial pressures is therefore not justified thermodynamically.

Mechanics and Kinetics

It has been pointed out that high pressures and temperatures are not needed for hydrogen reduction of most metal solutions. Yet commercial operation in plants using hydrogen reduction employs temperatures of 140° to 200°C and pressures of 400 to 800 psig, of which at least half is H_2 partial pressure. These conditions, according to present knowledge, have been found justifiable in order to obtain commercial reduction rates. They are still far less than those used by earlier investigators.

For a useful process it is not enough merely to make metal from its solution at a reasonable rate. It is also necessary to get the metal product in a form suitable for discharge from the reduction autoclave, that is, as a sand-sized powder product rather than as plate on the walls of the autoclave or as pebbles. Obtaining metal in suitable form is also a kinetic rather than an equilibrium phenomenon.

It is readily admitted that at the present time no attempt to formulate a complete kinetic explanation of metal reduction can reasonably be contemplated. Furthermore, the diversity of reduction kinetics makes the experimental picture very complex to present in full. To maintain the introductory character of this article an illustrative discussion is presented of the mechanics and kinetics of the reduction process and semi-quantitative data on one system only, the reduction of $CuSO_4$ with H_2.

A. Effect of Reduction Conditions: Primary controllable variables in hydrogen reduction are hydrogen pressure, temperature, solution composition, amount and kind of seed metal added, and agitation employed.

Example: Reduction of $CuSO_4$: The investigation was carried out with dilute copper solution to approximate the conditions of the last compartments in a continuous reduction system where the rates will be least. Also, complications due to possible formation of nonmetallic precipitates are avoided. Preliminary experiments on reproducibility indicated that data can be reproduced to a fair degree if heat-up effects are ignored. Some seed (oxidized surface) is leached during the heat-up period and causes some difference in copper concentration at zero time. The ordinates in Figs. 5 to 8 indicate concentration of metal remaining to be reduced.

1. *Hydrogen Pressure:* Effect of variation of the hydrogen partial pressure is shown in Fig. 5. The straight line portion of the curves in Fig. 5 at the lower hydrogen partial pressure indicates that the reaction may be controlled by dissolving of hydrogen, but at the higher partial pressure this effect disappears.

2. *Effect of Temperature:* A similar set of experiments was run with fixed hydrogen pressure at various temperatures. Results are shown in Fig. 6. It may be noted that the rate is considerably increased with temperature.

3. *Effect of Acidity:* Also included in Fig. 6 are curves for reduction of solution with varying concentrations of acid initially present. The adverse effect of acid on the reduction equilibrium was noted above in the theoretical discussion. Rate of reduction is also reduced by increasing acid strength.

4. *Effect of Sulfate:* Presence of sulfate, on the other hand, $(NH_4)_2SO_4$ or $FeSO_4$, for example, increased the rate considerably, as shown in Fig. 7.

5. *Effect of Seed:* In all the experiments of this section 80 gpl of copper seed was present. Seed used in these experiments was produced by disproportionation of an ammoniacal cuprous sulfate solution by acidification. Under these conditions the copper powder product consists mostly of octahedral crystals as illustrated in Fig. 9. Increase in copper powder hardly increases the rate of reduction. This is in marked contrast to conditions in nickel or cobalt reduction, Fig. 8.

6. *Effect of Agitation:* It has been mentioned that certain reduction tests gave a slow rate because they were diffusion-controlled. It will be shown below that the use of seed is frequently desirable to activate H_2 or to facilitate growth. To avoid diffusion-controlled reactions and to suspend the seed metal powder, adequate agitation is needed. In general this is intense agitation designed to keep the solids suspended and to maintain a good gas-liquid transfer. This is important for reactions carried out in expensive pressure vessels where economic considerations require that reaction time be maintained at a minimum. Although above tests on $CuSO_4$ were all carried out with the same agitation, experimental reduction data should always be taken as relative, and unfortunately comparison and scale-up of agitation, particularly for such complex reaction systems, are still far from being fully understood.

B. General Mechanism: Figs. 9 to 16 indicate the nature and variety of the powder product obtained from reductions. It may be noted that the usual metal product is highly agglomerated, but well formed crystalline metal particles can be obtained.

Metal preparation by reduction of the metal ion from solution with H_2 is composed of nucleation, growth, and agglomeration phenomena. Nucleation is the homogeneous formation from the metal ions of the smallest stable particle of metal. Growth for the present purposes is the atom-by-atom deposition of metal onto a solid surface, whether a metal or foreign surface. Agglomeration is the clustering of two or more metal particles to form an aggregate which can be permanently cemented together by an overgrowth of metal.

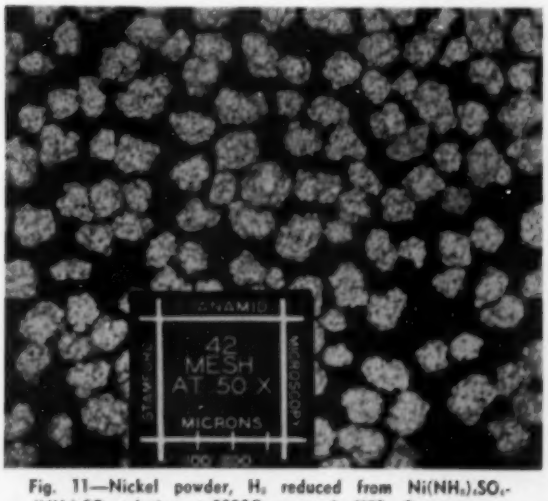

Fig. 11—Nickel powder, H_2 reduced from $Ni(NH_3)_2SO_4$-$(NH_4)_2SO_4$ solution at 200°C, on seed. X50. For cross section see Fig. 12.

- The Chemico metals technique is proving its worth, as several plants are now operating close to or at capacity and consistently produce copper, nickel, and cobalt of commercially attractive purity at competitive costs.
- Having met contract performance, American Cyanamid Co. turned over operation of the Calera plant of Howe Sound Mining Co. to its owners on Dec. 8, 1955. A practical commercial operation, the plant is reported to have produced in March 250,000 lb of cobalt containing 3 to 4 pct Ni.
- Sherritt Gordon operations for 1955, including an ammonia leach of the company's own design, have already been reported as satisfactory and are said to have exceeded plant capacity. Twenty-five tons of nickel per day and some cobalt are now produced.
- National Lead's Fredericktown plant for recovery of copper, nickel, and cobalt from 50 tpd of low grade sulfide concentrate is said to operate at about two-thirds capacity.
- The Whitaker Metals Co. scrap copper leach plant has recovered as much as 240,000 lb per month of a premium product for the copper metal powder market. The plant is now operated by Fluor Corp. as a semi-pilot plant.
- Freeport Sulphur Co.'s demonstration plant for recovery of nickel and cobalt from 50 tons of Cuban laterite per day is reported to be operating satisfactorily at full capacity.
- Chemical Construction Corp., which did all the early research and development work and the design and construction of the plant, was sold by American Cyanamid Co. to Ebasco in April 1956. Negotiations are in progress for sale of the metals processes patents.

Earlier kinetic and electron microscopy study on gold sols by Turkevich et al. should be mentioned.[29]

Qualitative reducibility of a given metal ion naturally depends on the chemical nature of the metal ion, that is, on whether the ion is complexed by water, chloride, ammonia, or even organic molecules. Cadmium acetate, for instance, appears to reduce much more rapidly from an alcohol solution than from an ammine complex in an aqueous solvent.

Metallic cadmium was precipitated in small yields by H_2 reduction from several ammine sulfate solutions at 224°C and 1200 psi total pressure. A particularly pure metal product, however, is obtained in good yield, 65 and 80 pct, respectively, from a 0.5 *M* cadmium acetate solution in methanol or isopropanol at 200°C and 500 psi H_2 partial pressure.

In this connection it may be noted that cobalt ammine complexes often are markedly stable from a kinetic viewpoint.[30]

1. *Nucleation:* Homogeneous nucleation of metal particles is usually difficult compared to their growth. For example, reduction of copper, nickel, and cobalt from ammoniacal solution will usually nucleate only with difficulty but will readily grow additional metal onto metal surfaces already present. On the other hand, reduction of copper from acidic solution nucleates (and grows) readily, presumably merely because a larger number of metal ions are available in acidic solution. In ammoniacal solution the problem is therefore one of supplying a metal surface, or seed, for growth. In early work at Chemical Construction Corp. this was accomplished by grinding the oversize metal product. However, grinding is cumbersome and a source of contamination of the product; also it is not possible to restore the original internal surface of the early precipitates. It soon became evident that improved means of nucleation had to be developed, such as: 1) use of a nongaseous

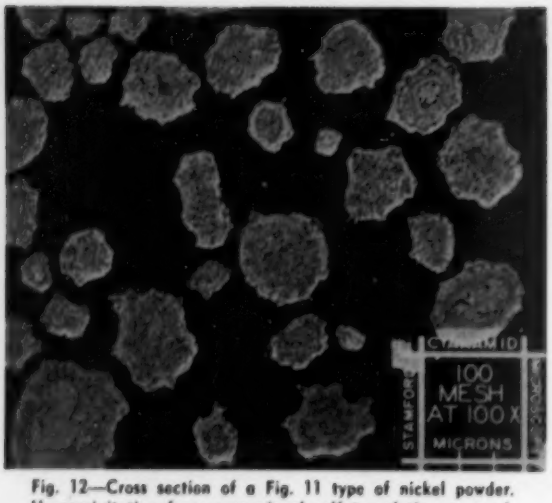

Fig. 12—Cross section of a Fig. 11 type of nickel powder. H_2 precipitation from ammoniacal sulfate solution on seed. Note growth by agglomeration and deposition (onion structure). X100.

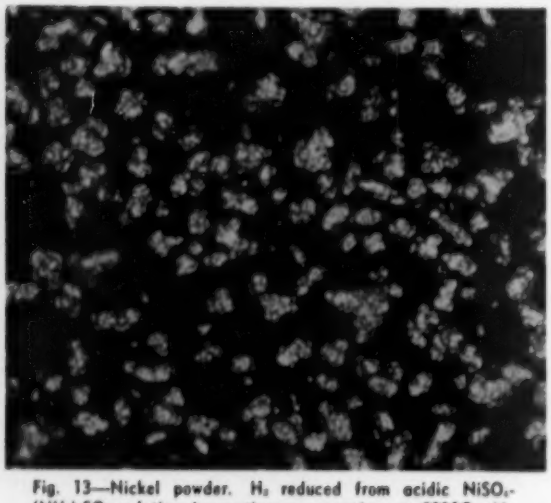

Fig. 13—Nickel powder. H_2 reduced from acidic $NiSO_4$-$(NH_4)_2SO_4$ solution in continuous operation at 200°C. No seed. X75.

Fig. 14—Electron micrograph of nickel needles from a powder obtained by H_2 reduction of $NiSO_4$-H_2SO_4 solution at 200°C. Area shown is approximately 3μ wide.

reducing agent (e.g., hypophosphite or hydrazine) to produce a fine metal colloid from a minor portion of the solution; 2) performing a self-nucleating acid reduction on a minor part of the solution; or 3) use of solid compounds that catalyze the growth of metal by H_2 reduction.

Nickel and cobalt metal, certain oxides, and various other hydrogen activating compounds are used industrially for hydrogenation of unsaturated organic compounds. It appears plausible that such compounds would also catalyze a metal reduction. Heterogeneous catalysis was indeed found to be important for the nucleation of metal in reduction by H_2, and it happens in the case of nickel and cobalt precipitation that the reduction products themselves are among a number of possible hydrogenation catalysts. On the homogeneous catalytic activation of H_2 by Ag^+, Hg^{++}, or Cu^{++}, see Halpern.[28]

It should be noted that it is often difficult to decide whether in a given reduction run the metal had truly nucleated in the bulk of the solution or whether reduction was initiated on accidental impurities such as dirt or oil from the stuffing box.

2. Growth: Crystal growth is a sequence of dissolution, bulk diffusion, surface adsorption, surface diffusion, chemical reaction, deposition, desorption, and other reactions, and the kinetics of growth is concerned with the kinetics of that sequence. Gross reaction rate during a reduction should somehow depend, in a first approximation, upon the effective metal ion concentration, the H_2 molecule concentration (in solution), the metal surface area, and temperature. Other effects of lesser immediate interest include the variation in growth rate between different crystal faces, activity coefficients, and poisoning of the metal surface by adsorption.

Quantitative studies of growth kinetics are severely obscured by agglomeration of particles during reduction, which makes it difficult to determine surface areas. Also, as described above in reduction from ammoniacal solution the varying NH_3: M^{++} and NH_4^+ mol ratios during reduction cause a varying effective concentration of metal ion. For the present purpose, however, it is evident that the amount of surface area is important, see Fig. 8, and that the nucleation reaction noted above should give a high area of metal. The structure of the nickel powder shown in Fig. 12 indicates growth by deposition (onion structure) and agglomeration.

Even the mechanism by which crystal growth occurs is not clear. Recent investigations increasingly support a screw dislocation mechanism for crystal growth rather than the classical two-dimensional, surface nucleation viewpoint. In metal reduction, a filamental type of crystal growth has often been observed which appears to be the result of a dislocation mechanism, see Fig. 14 and Ref. 30.

Since cobalt and nickel metal are ferromagnetic, it would perhaps be expected that growth of these metals in a magnetic field would affect the growth kinetics or at least the magnetic properties of the resulting metal crystals. In the laboratory of Chemical Construction Corp. W. G. Courtney has observed an increase in the coercive force of these metals when they were prepared in a magnetic field by reduction with hypophosphite. However, he could observe no effect on the coercive force in reduction by H_2. This suggests a qualitatively different growth mechanism with the two different reducing agents.[31]

3. Agglomeration: From colloid chemistry it is known that, unless stabilized, solid particles in the

size range of 0.1 to 5 μ agglomerate. A particle may agglomerate either to another particle or to the autoclave wall. According to theory, the tendency for either type of agglomeration to occur depends upon particle concentrations, surface area of the wall, type and degree of adsorption of ions onto the surfaces of the particles and walls, degree of agitation, and many other factors. Appropriate data at elevated temperature are again lacking; even at room temperature the agglomeration process is not well understood.

Agglomeration is doubly undesirable. Agglomeration between particles decreases the total metal surface area and thus the reduction rate. Agglomeration to the wall, or wall plating, Fig. 15, not only removes metal from the immediate production but also requires equipment downtime for its periodic removal. After plating starts there is a catalytically active metal surface on which additional metal will be easily deposited, unless a much larger surface of metal in the bulk solution is provided in the form of metal seed particles. On the other hand, agglomeration, in cases of slow growth, is the only mechanism available to consolidate small particles to form metal powder coarse enough to prevent its redissolution in subsequent washing steps or dusting in drying operations.

Two observations from separate and dissimilar studies will be given to illustrate the agglomeration process. Successive samples of a $NiSO_4$ reduction conducted at slow rate showed that in the first sample the nonagglomerated particles have a size range of 0.1 to 0.4 μ (by electron microscope). In the second sample taken a few minutes later, the particles are either 0.5 to 0.8 μ or are agglomerates of 2 to 5 μ total diameter.

An autoclave with glass walls (a sight glass) was used to reduce copper from an ammine carbonate solution with CO. As the temperature was increased toward 120°C, a Tyndall effect was noted. A few minutes later, presumably after a critical particle size had been reached, a sudden coagulation occurred which formed a few visible agglomerates in the bulk of the solution and rapidly coated the glass wall with copper foil.

Purity

Impurities in a metal reduction can arise from: 1) superficial entrapments of solid salts or solution during agglomeration of metal particles, 2) entrapment by agglomeration but followed by an overgrowth of additional metal, or 3) an intimate mixture of impurity and metal atoms arising from incorporation of impurity atoms during the atom-by-atom growth of metal during reduction.

Reduction from insufficiently buffered ammoniacal solution at excessive temperatures may lead to impure powders. The fluffy basic metal compounds that precipitate on heating up are likely to retain impurities, and since such compounds normally reduce directly by solid-gas reaction without being redissolved as reduction proceeds, contamination by carbon from carbonate or sulfur from sulfate solution, oxygen, or contained impurities is in certain cases expected.

It is also possible at temperatures near 250°C to reduce sulfate to sulfide with hydrogen. Another reason for using control of environment to facilitate reduction is that such high temperatures are not needed.

Pure metals can also be obtained from impure solutions. Impurities such as iron, zinc, aluminum, and magnesium are not reduced to metal and will not contaminate the metal powder if kept in solution. Copper powder containing 0.005 pct Fe has been produced in acid reduction from solution containing up to 15 gpl Fe^{++}. Even cobalt and nickel, if present, do not contaminate copper if the copper

Fig. 15—Nickel foil and agglomerates as stripped from the walls of the autoclave. Reduction of an ammoniacal nickel solution at 175°C with H_2. No seed was provided. Compare with Fig. 13, nickel from an equally unseeded reduction, but from a self-nucleating acid system. X3.

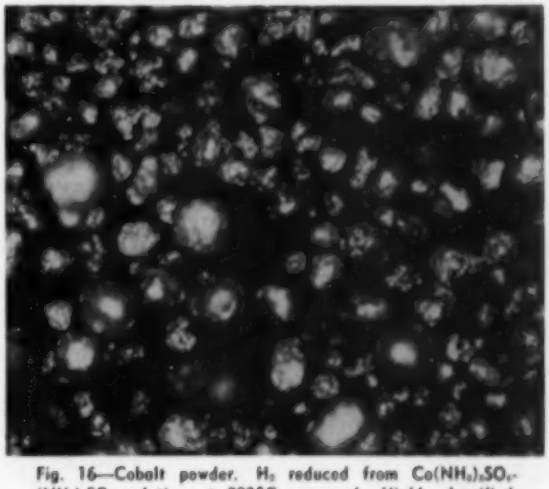

Fig. 16—Cobalt powder. H_2 reduced from $Co(NH_3)_4SO_4$-$(NH_4)_2SO_4$ solution at 200°C, on seed. Highly densified. Note how in contrast to Fig. 11 surface of agglomerate is smoothed out by subsequent metal deposition. X75.

Table III. Analyses of Copper, Nickel, and Cobalt Powders

	No. 1 Cu	No. 2 Ni	No. 3 Ni	No. 4 Ni	No. 5 Co	No. 6 Co	No. 7 Co
Cu	99.948	0.01	nil	0.037	nil	0.01	0.02
Ni	nil	99.85*	99.72	99.74	0.15	0.54	3.7*
Co	nil	0.1	0.24	0.15	99.6	98.49	95.7
Fe	0.007	0.01	0.025	0.037	0.20	0.08	0.07
Zn	nil		nil				nil
Sn	0.003						
Pb	nil						
S	nil	0.01	0.001	0.015	0.02	0.08	0.05
C	0.011						
H_2 loss	0.008						

* No nickel-copper separation carried out at this plant.
Nos. 1-3 and 5 are products prepared in different piloting operations[1,2]
Nos. 4, 6, and 7 are plant products.

is reduced from solution with a pH of less than 1 where, as shown by Fig. 3, cobalt or nickel are not reduced at equilibrium conditions. In one commercial operation copper is reduced from a solution richer in nickel and cobalt than in copper and the powder produced analyzes less than 0.01 pct of each, nickel and cobalt.

Copper analyzing 99.99 pct has been produced from sulfate and from carbonate solutions. In Table III a few product analyses from various industrial and larger piloting operations are given.

In general each problem is different and any required purity can be obtained by a combination of controlled reduction with prepurification and subsequent washing.

Apparatus

Experimental data presented here have been obtained during the course of a program of applied research extending over a period of several years. Autoclaves used include rocker bombs, 1-gal autoclaves heated with Dowtherm jackets or with gas and usually equipped with a liner. Temperatures are usually controlled within 2 to 5°C. A few experiments have been run in 10 and 25-gal autoclaves. These are heated with a steam coil.

Although the air in the clave could be removed by heating and venting, because many solutions may contain ammonia, it has been the standard practice to purge the clave with nitrogen before heating.

The pressure vessels were usually provided with sample lines, some having porous stainless steel or tantalum micro filters, for sampling at operating temperature and pressure into sample bombs. In experiments of qualitative nature, the pressure vessels were cooled before discharging, although some redissolution of metal may have resulted.

Considerable quantities of reducing gases are consumed in these reduction reactions. In view of their low solubility, maintenance of good absorption and diffusion rates by suitable stirring is important.[20] Various types of agitation have been employed.

Most of the various physical and chemical reactions that occur during a reduction appear to be greatly affected by what may be called the hydrodynamics or the stirring pattern, which in turn are largely functions of the geometry of the stirrer and vessel, of the stirring rate, of the composition of the solution, and of the reaction conditions. Efficient dispersion of solids (seed metal, catalysts and newly precipitated metal) is as important as good gas distribution. Excessive centrifugal action should be avoided in order not to plaster the autoclave walls with growing metal particles. Rough surfaces are undesirable, as they become coated much faster.

Precipitated powdered metal was usually washed with water, alcohol, and acetone or ether and then dried on a steam bath. In some tests the powders were dried with hydrogen at elevated temperatures.

Acknowledgment

Thanks are due to the management of Chemical Construction Corp. for permission to publish this paper and particularly to E. S. Roberts, chief engineer, and P. J. McGauley, head of the metals div., for their constant support of this project.

For numerous discussions and great help the writer is indebted to his coworkers at Chemico and Sherritt Gordon Mines Ltd., particularly to J. O. Dasher and W. G. Courtney, and to T. G. Rochow and his coworkers of American Cyanamid Co. who prepared the micrographs.

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Discussion of this paper sent (3 copies) to AIME by Aug. 31, 1956, will appear in MINING ENGINEERING and in AIME *Transactions*, Vol. 205, 1956.

aime news

AIME Adopts New Seal

With the adoption of the new name of the Institute, including the word *Petroleum*, it has seemed desirable to design a new emblem, shown on p. 449 of this issue. The crossed hammers have been the symbol of mining and metallurgy for hundreds of years, and long before oil was discovered in quantity. The new design includes something characteristic of the petroleum industry as well. The new emblems are expected to be available early in the summer, at which time members will be advised. The price will probably be the same as for the former pin, $4 plus 10 pct tax. The emblem will be available as a pin, lapel button, or tie clasp.

Award Committees

Funds are sufficient to finance all of the usual AIME Awards to be presented at the Annual Meeting in New Orleans Feb. 24 to 28, 1957, except the Ramsay Medal for achievement in coal mining.

Award committees have been named, and the chairmen will be glad to hear from any AIME members who wish to suggest the name of a candidate worthy for consideration by the appropriate committee. Their names follow: W. A. Dean, committee chairman, James Douglas Medal for distinguished achievement in nonferrous metallurgy; C. E. Weed, chairman, W. L. Saunders Medal for mining; R. W. French, chairman, Anthony F. Lucas Medal for petroleum; M. L. Haider, chairman, Charles F. Rand Medal for mining administration; C. Y. Garber, chairman, Robert H. Richards Award for mineral beneficiation; J. S. Marsh, chairman, Benjamin F. Fairless Award for iron and steel; C. M. Squarcy, chairman, J. E. Johnson Jr., Award for iron and steel (recipients under 40); J. C. Calhoun Jr., division chairman, Mineral Industry Education Award; K. L. Cook, division chairman, Jackling Award for mining, geology, or geophysics.

The following awards have also been authorized for best papers: Robert W. Hunt Medal, Mathewson Medal, Extractive Metallurgy Division Award, and the Rossiter W. Raymond Award.

AIME Northeastern Mining Branch Conference

Hershey, Pa. November 8-9-10

Comprehensive branchwide technical program with nonconflicting field trips is planned.

Host: Lehigh Valley Section, D. S. Lyons, Section Chairman; W. B. Stephenson, General Conference Chairman.

Look for further details in June issue of Mining Engineering.

ECPD Submits New Engineering Definition

A new definition of engineering was submitted by the Engineers Council for Professional Development, and approved by the AIME Board of Directors at their February meeting. It states: "Engineering is the profession in which a knowledge of the mathematical and physical sciences gained through study, practice, and experience is applied with judgment to the known materials and forces of nature for the comfort, health, safety, and welfare of mankind."

Local Section Formed

A new local section has been carved out of territory formerly in the area of the Delta Section. It will be known as the **Evangeline Section,** and comprises the following Louisiana parishes: Vermillion, Jefferson Davis, Allen, Evangeline, Acadia, Lafayette, St. Landry, Pointe Coupee, St. Martin, West Baton Rouge, Iberville, and Iberia. Chairman of the new section is Drew Cornell. D. D. Jones is Secretary.

Arizona Students Active

The Arizona Student Chapter reports a varied school year program. As well as participating in traditional activities, it operates the Mines Building during exhibit days, giving conducted tours of displays. Several leading men from industry have addressed the group, and at a recent meeting a USBM member presented a film on the electron microscope.

The tutoring program initiated this year is pointed to with pride. First limited to helping younger students in basic courses, assistance has been extended to all Mines College students, even those in senior courses, and results are encouraging.

Long Range Planning Committee Report

The Long Range Planning Committee report has been received by the Board of Directors for consideration at the June Board of Directors meeting. Action on the report has not been taken.

The report will be published in the June issue of MINING ENGINEERING. Copies of the report are now available upon request from AIME headquarters.

It will cost you 2c to help us help you . . .

The office of the Mining Branch Secretary would like to revise and bring up to date their files on members who serve as **full time or part time consultants** to the Mining Industry. This information is needed to properly answer the many requests which are directed to our attention by persons writing in from all parts of the world. If you are interested in having your name listed and kept in this up-to-date file, kindly send your name, latest permanent address, and special field of endeavor or service as a consultant on a "Two Penny" Post Card to: **Arnold Buzzalini, Mining Branch, AIME, 29 West 39th Street, New York 18, New York.** . . . Closing date for the first list will be July 1, 1956. It is planned that this listing will be revised periodically in the future.

Around the Sections

• Sixty-one members and eight guests attended the March 6 dinner-meeting of the **Washington, D. C., Section** at the Cosmos Club. Speaker for the evening was John Garrett, executive secretary, coordinating committee on materials, U. S. Dept. of Defense. Mr. Garrett discussed titanium metal, its properties and uses, and the role of the Dept. of Defense in its development and future. In addition, a 16-mm sound film, *The Ninth Element*, depicting mining and processing of ilmenite by the National Lead Co., was shown.

Members who attended the meeting on April 3 at the Cosmos Club, heard Ulysses N. Staebler, chief, AEC, civilian power reactors branch, division of research development. Mr. Staebler discussed *Power From Nuclear Reactors*.

On April 23, the Section was invited to a meeting of the District of Columbia Society of Professional Engineers at the PEPCO Auditorium. The invitation was extended by the chairman of the Society, Wallace D. Barlow, who is also an AIME member. An Eastman color film, *Mining For Nickel*, released by International Nickel Co., was shown. In addition, Herman B. Director, formerly advisor to the Secretary of Commerce on Industrial Materials, was guest speaker.

The May meeting of the Section, originally scheduled for May 1, has been postponed until May 8. At that time, Simon Strauss, vice president of American Smelting & Refining Co., who recently returned from abroad, will discuss mining in Europe. On the agenda for June 5, is a field trip to the Baltimore plant of Kaiser Aluminum & Chemical Corp.

• The **Reno, Nev., Subsection** met March 9 at El Cortez Hotel. John N. Butler reported on the 1956 AIME Annual Meeting which he attended as a Nevada Section delegate.

At the March 6 meeting of the Crucible Club, the Mackay School of Mines, University of Nevada, Student Chapter, student Karel Havelik spoke on his personal experiences in the slave labor uranium mines of Czechoslovakia.

• On March 7 the **Montana Section** met in Butte at the Montana Power Auditorium. Charles S. Burriss, plant manager of the Ideal Cement Co. at Trident, Montana, gave a talk on *Cement Manufacturing from Quarry to Finished Product*. The talk was illustrated by color motion pictures and slides.

A recent highlight of the Section's activities was the visit on April 9 of C. E. Reistle, Jr., AIME President; E. O. Kirkendall, Secretary of the Institute; and R. E. O'Brien, Field Secretary from Salt Lake City. The dinner in their honor was held at the Hotel Finlen in Butte. Principal speaker was President Reistle who discussed phases of the petroleum industry, in which he is recognized as an international authority.

The following members were elected for 1956: J. R. Van Pelt, chairman; T. K. Graham, vice chairman; Clifford J. Hicks, secretary-treasurer. The executive committee will also include Stanley M. Lane, Edward P. Shea, and Godfrey Howard, president, Anderson-Carlisle Soc. Committee chairmen are: Fred E. Burnet, membership; R. G. Bowman, program; Edward I. Renouard, Jr., student awards; and Clifford F. Milwick, nominating committee.

A joint AIME-Anderson-Carlisle Student Soc. Annual Get-Together was held on April 25 at the Museum Hall of the Montana School of Mines in Butte. The program, a student symposium, included papers on *Geochemical Prospecting* by T. Eyde, *Zone Refining of Metals* by E. Westerman, *Formation Fractoring as an Aid in Increasing Oil Production* by L. McCarthy, and *Streaming Methods* by A. Mular.

• The Metals and Mining Groups of the **Philadelphia Section** held a joint meeting on March 29, 1956, at the Lenape Club. The meeting was preceded by a cocktail hour.

W. C. Mearns, of the International Nickel Co., presented a 45-min color movie on *Mining for Nickel*. This film presented a comprehensive view of the modern techniques employed in the mining of nickel ores at Sudbury. This film started with illustrations of modern exploration methods, and then showed clearly, six different techniques for removing the ores.

A very enjoyable and informative question period followed the film.

• The **Colorado Section** met January 19 at the University Club in Denver. Max Gelwix, head of the planning department, Climax Molybdenum Co. explained testing methods, equipment selection, and project planning at his company.

The Board of Directors was asked to study the problem of holding the 1957 Rocky Mountain Minerals Conference in Denver and to submit their recommendations.

• The Annual Meeting of the **Colorado MBD Subsection** to be held jointly with the **Pike's Peak Section,** will take place at the Broadmoor Hotel in Colorado Springs on May 19. Elections, the annual business meeting and report are on the agenda, as well as a dinner-dance. An excellent technical program has been arranged by John Ferguson and will feature five papers on the various phases of MBD work. The ladies are invited to attend a special program organized for them by the Pike's Peak Section.

• Ward Minkler, assistant manager for market development, Titanium Metals Corp. of America, addressed the **New York Section** on March 29, 1956, speaking on the problems of the titanium industry today, and titanium's future. This was a joint meeting with the Section's Physical Metallurgy Group. The dinner-meeting was held at the Mining Club.

Mr. Minkler summarized the sources of titanium ore and the methods used in preparation of titanium sponge. A lively question and answer period followed the lecture.

The March meeting was the occasion for John H. Ffolliott to relinquish the chairmanship of the New York Section to Herbert Z. Stuart, incoming chairman.

The Section will hold a Hob-Nob Meeting in association with the powder metallurgists in this area and the N. Y. Physical Metallurgy Group. Scheduled for May 16, this dinner-meeting will be held instead, on Thursday, May 24 in the Grill Room of the Brass Rail Restaurant on Fifth Ave. at 43 St. in N. Y.

Featured speaker is N. J. Grant, Professor of Metallurgy, Massachusetts Institute of Technology, Cambridge, Mass. He will discuss *Dispersed Phase Hardening*, reviewing the development and performance of the aluminum and aluminum-oxide alloys of the SAP variety, particularly the theory behind high strength values achieved. Prof. Grant will describe the problems to be solved for successful alloys of the metal-metal oxide type to be practical.

• On March 9 the **St. Louis Section** met at the York Hotel to hear a study presented by Ross R. Heinrich, professor of geophysics and geophysical engineering at St. Louis University. The topic was *Some Objectives of the Analysis of Ground Vibration Records*. Dr. Heinrich's interests lie with the relation of earthquakes in mid-continental U. S. to regional structure, and the characteristics of the propagation of elastic waves in the rocks near the earth's surface.

(Continued on page 552)

Round Up Time

CHUCK "AIMEABLE SAYS...

We're Out To Get The Strays And Dogies...

A.I.M.E. MEMBERSHIP... IS A MUST.

~ A.I.M.E. IS YOUR PROFESSIONAL SOCIETY ~

WHEN round up time comes around, out in the cattle country, all hands pitch in and the work is quickly and effectively completed. When all the cattle are in, those already branded and in good shape are cut out, and particular attention is paid to the strays and the dogies.

We've had our round-up in New York. But what about the strays and the dogies?

The strays are those members of our profession, our associates who haven't been branded as members of AIME. Everyone of us, by looking around, can find many of these who could be brought into the fold. By giving us their names and addresses, you can help the Membership Committee in their work. While you are at it, you can do a little selling on what AIME has done for you and what it means to you, just to steer them to the corral.

The dogies are our recent graduates who, on January 1st, were automatically transferred to Junior Membership. There will be a new crop soon; graduation day 1956 is fast approaching. There are also Junior Members who reached the age where they are transferred to Associate Membership. Why not extend to them the hand of friendship which has characterized our profession. We need these younger men; they are the leaders of the future.

So why not imitate **Mr. AIMEable,** the *Symbol* of your Membership Committee? Mount your trusty *Cayuse*, get out your lariat, and help us rope them in. This is round up time.

*C. E. Golson**

* *Chairman, AIME National Membership Committee.*

INDUSTRIAL MINERALS DIVISION

NEWSLETTER

Dear Members of the
Industrial Minerals Division:

At the annual Executive Committee meeting of the Division held in New York several months ago, it was decided to use a portion of a column in MINING ENGINEERING for an IndMD newsletter four times a year. This is the first of a new quarterly series of newsletters, and any special items of interest should be addressed to me, Robert M. Grogan, Secretary—Industrial Minerals Div., AIME, 6013 du Pont Building, Wilmington 98, Del.

Twenty-two committee chairmen and past chairmen attended the annual Executive Committee meeting in February, and this group plans to meet again in early May. Sandford S. Cole and Raymond B. Ladoo reported our Division will assist in the Mining Branch Northeastern Regional meeting scheduled for November 8th, 9th, and 10th at Hershey, Pa. The Division will have no separate regional meeting this year. It is hoped to favor the AIME Lehigh Valley Section, who are the local sponsors for this meeting, with a large attendance, and expect an interesting program will await those who attend. Chairman for this regional meeting is W. B. Stephenson, The Allen Sherman Hoff Pump Co., 259 E. Lancaster Ave., Wynnewood, Pa.

Vernon E. Scheid, Pacific Southwestern vice-chairman announced a southwestern Minerals Conference is being planned for Reno, Nev., for the fall of 1956, and invited our Division to participate by contributing to the program and through member attendance. It is proposed to hold this meeting annually in San Francisco, Los Angeles, and Reno on a rotational basis, following a plan similar to that now being established in the Northeast.

Other items discussed at the Executive meeting include the new Division abbreviation IndMD, this new form of a quarterly newsletter, committee on definitions to advise the American Standards Assn. on industrial minerals materials and the revision of the 1949 volume *Industrial Minerals and Rocks*. Since this meeting we have learned the Mudd Fund has adequate resources to take care of the cost of revision.

Commodity Committee Chairmen

Wilbur T. Stuart	Industrial Waters
Walter E. Trauffer	Cement, Lime, & Gypsum
Pauline Moyd	Ceramic Raw Materials
G. D. Emigh	Chemical Raw Materials
Ralph A. Fletcher	Dimension Stone & Slate
Arthur L. Hall	Fillers, Fibers, & Pigments
Thomas D. Murphy	Mineral Aggregates
George F. Pettinos, Jr.	Special Sands & Abrasives
David M. Larrabee	Rare Metals
Thomas E. Gillingham, Jr.	Program Chairman—384 Morris Ave., Mountain Lakes, N. J.

Industrial Minerals Division

Chairman of Division	Raymond B. Ladoo—42 Huntington Rd., Newton 58, Mass.
Secretary-Treasurer	Robert M. Grogan—see address above
Eastern Vice Chairman	John G. Broughton—N. Y. State Science Service, Albany 1
Southeastern Vice Chairman	J. L. Weaver—American Cyanamid Co., Brewster, Fla.
Mid-Continent Vice Chairman	John C. Frye—Illinois State Geological Survey, Urbana, Ill.
Rocky Mountain Vice Chairman	J. J. Norton—Mineral Deposits Branch, U. S. Geological Survey Building 25, Federal Center, Denver 14
Pacific Southwestern Vice Chairman	V. E. Scheid—Mackay School of Mines, Reno, Nev.
Pacific Northwestern Vice Chairman	M. T. Huntting—Washington Div. of Mines & Geology, Olympia, Wash.
Canadian Vice Chairman	Norman B. Davis—Victoria Building, Ottawa, Ont., Canada

Executive Committee

To 1957—Carl F. Clausen, Portland Cement Assn.
33 West Grand Ave., Chicago 10

George D. Dub, Cyprus Mines Corp.
523 West 6th St., Los Angeles 14

Richard W. Smith, Natural Resources Dept.
U. S. Chamber of Commerce, Washington 6, D. C.

To 1958—Thomas L. Kesler, Box 31,
Kings Mountain, N. C.

R. J. Lund (filling unexpired term of R. B. Ladoo),
Battelle Memorial Institute, Columbus 1, Ohio

A. O. Bartell (filling unexpired term of J. G. Broughton),
308 Woodlark Building, Portland 5, Ore.

To 1959—John H. Weitz, Independent Explosives Co.,
455 Leader Building, Cleveland 14

R. H. Jahns, California Institute of Technology,
Pasadena, Calif.

T. D. Murphy, Pennsylvania Glass Sand Corp.
Berkley Springs, W. Va.

We hope many of you have already sent in your suggestions and offers to present papers at the coming regional and annual meetings. Our next newsletter will appear in this column in the August issue.—*R.M.G.*

Around the Sections, Cont'd.

• The **San Francisco Section** held a joint annual dinner dance with the ladies' auxiliary at the Terrace Room of the Fairmont Hotel on March 17. Rod McCauley's orchestra provided dance music, and the intermission periods were filled with diverting entertainment.

Guest speaker for the meeting on April 13 was Carl E. Reistle, Jr., AIME President. The Engineers' Club was the scene for this President's meeting. Mr. Reistle, who is executive vice president in charge of production for Humble Oil Co., was accompanied by Ernest Kirkendall, Institute Secretary, and Roy O'Brien, Field Secretary.

A new 30 min film, *Million Dollar Drill Holes*, is now available for distribution. The 16-mm film shows the complete process of drilling for uranium and many colorful pictures of the Colorado Plateau.

It should be of particular interest to miners, machine supply houses, and bankers, and will be furnished free of charge to organizations. Address requests to Charles A. Steen, president, Moab Drilling Co., Moab, Utah, including three dates for showing the film so that the company can select one to fit their schedule.

This film is complementary to *This is Mi Vida*, a 45-min 16-mm film on the production of uranium, which was released by Charles Steen's Utex Exploration Co. about a year ago. This film, which is still available, has been viewed by some 50,000 people in the U. S., England, France, Australia, and Mexico.

Diamond Drillers Meet In Canada

An International Diamond Drilling Conference sponsored by the Canadian Diamond Drilling Assn. will be held May 31 through June 2, 1956, at the Royal York Hotel in Toronto. In conjunction with this conference, there will also be the thirteenth annual convention of the Canadian Diamond Drilling Assn. and the spring meeting of the Diamond Core Drill Mfrs. Assn. (U.S.A.).

The conference will feature technical papers on diamond drilling by men from the U. S., Canada, Australia, and S. Africa. Of particular interest will be two papers: *Screw Feed Versus Hydraulic Feed as Used in Canadian Drilling* by Perry Hall, president, Boyles Bros. Drilling Co., Vancouver; and the same topic as applied to U. S. drilling by M. L. Tucker, Chicago Pneumatic Tool Co., New York.

Directors Meet

Future meetings of the Board of Directors or the Executive Committee will not be held monthly, as they have been in the past. Election of new members and other matters which require prompt action will be conducted by mail. The dates decided upon for Board meetings are: June 20 and 21, and Nov. 21, 1956, both in New York; two meetings in New Orleans are scheduled for Feb. 24 and Feb. 26, 1957.

The Executive and Finance Committees will meet jointly with the Treasurer and Secretary of the Institute in New York on the following dates: Sept. 20 and Dec. 19, 1956.

At the first meeting of the new Board of Directors, Feb. 21, 1956, the following were named for the offices and committees indicated: Gail F. Moulton to succeed himself as Treasurer and Controller for a one-year term; William J. Masson to continue as Assistant Controller; Ernest Kirkendall to succeed himself as Secretary for one year; Edward H. Robie reappointed as Secretary Emeritus, until Nov. 30, 1956.

The Executive Committee is headed by C. E. Reistle, Jr., as chairman; H. DeWitt Smith, vice chairman, and T. B. Counselman, Grover J. Holt, and A. W. Thornton. The Finance Committee will be led by Andrew Fletcher, chairman; R. B. Caples, and C. R. Dodson.

Chairmen of the various fund committees were named as follows: Endowment, W. W. Mein, Jr.; Endowment Fund X, G. F. Moulton; Rocky Mountain Income, Henry Krumb; Seeley W. Mudd Memorial Fund, John M. Lovejoy; Gemmell Memorial Fund, Louis S. Cates; Henry L. Doherty Memorial, John M. Lovejoy.

Chairmen of the Awards Committees are: Robert H. Richards Award, C. Y. Garber; Rand Foundation Award, M. L. Haider; Douglas Metallurgical Medal, C. C. Long; Saunders Mining Medal, C. E. Weed; Lucas Petroleum Medal, R. W. French; Erskine Ramsay Coal Medal, L. C. Campbell; Student Prize Paper Awards, David H. Beetem; Benjamin F. Fairless Award, J. S. Marsh.

AIME Committees will be guided by the following chairmen: Honorary Memberships, Andrew Fletcher; Admissions, R. B. Caples (chairman), and F. A. Ayer (vice chairman); Members' Conduct, W. M. Peirce; AIME Pension Trust, G. F. Moulton.

Chairmen of the AIME intersociety committees are: Library, Frank T. Sisco; Cooperation with the CIM and MSNS, P. J. Shenon; Research, F. B. Foley.

Lack of facilities will preclude any exhibits by industrial companies at the Annual Meeting in New Orleans next February.

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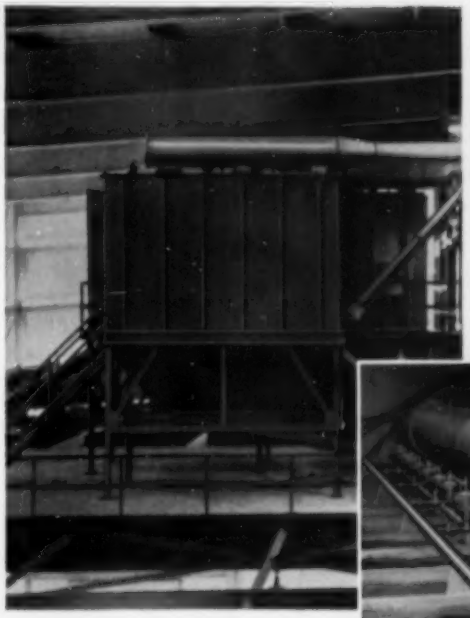

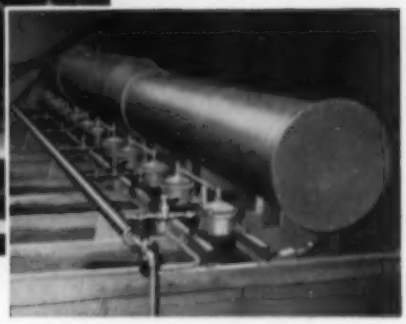

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PERSONALS

Henry H. Hausner is general manager, nuclear engineering div., Penn-Texas Corp., N. Y. He will also act as consultant for Westinghouse Electric Co.'s atomic power div., Nuclear Development Corp., Glenn Martin Corp., Argonne National Laboratories etc. At Penn-Texas, Dr. Hausner will advise on the application of nuclear energy for industrial purposes. Formerly manager of engineering of Sylvania Electric Products' atomic energy division, he has also served as professor, New York

HENRY H. HAUSNER

University School of Engineering and is currently teaching at Brooklyn Polytechnic Institute. International authority on powder metallurgy, Dr. Hausner wrote the first technical book on this subject in 1947, and is also the author of *Materials for Nuclear Power Reactors*, published last year. He was general chairman of the recent symposium on *Powder Metallurgy in Atomic Energy* at the Philadelphia convention sponsored by AEC and ASM.

Philip E. McGarry is Research Investigator, Ore Dressing Dept., New Jersey Zinc Co., (of Pennsylvania) at Palmerton, Pa.

J. N. McClure is manager, North Carolina mining operations, Lithium Corp. of America Inc., Bessemer City, N. C.

Howard M. Fowler has opened a combination consulting and exploration office in Vancouver, and has available both scintillometer equipment and aircraft. Mr. Fowler is a professional engineer in Alaska and British Columbia.

Allan F. Matthews economic geologist for the U. S. International Cooperation Administration, has been transferred from Washington, D. C., to Bonn, Germany. He is program officer for U. S. aid, now largely limited to Berlin.

PHILIP D. WILSON

Philip D. Wilson is president and **Robert P. Koenig,** vice president of the Mining & Metallurgical Society of America for 1956. Mr. Wilson, a consulting mining engineer and geologist has been associated for some years with Lehman Bros. and the Lehman Corp. He is a director and member of AIME finance committee. Mr. Koenig, president of Cerro de Pasco Corp., and chairman of Circle Wire & Cable Corp., has also been prominent as a mining engineer in Peru and the U. S.

ROBERT P. KOENIG

Charles K. Leith received a citation for outstanding services from the AEC on Apr. 17, 1956, on his retirement as a member of the Combined Development Agency. The latter, is a joint U. K.-U. S.-Canadian organization established 1944 in Washington to procure and develop the production of uranium and thorium supplies in such areas as the Belgian Congo, Australia, and Portugal. Dr. Leith has been one of the three U. S. representatives on the agency since its establishment. A noted geologist, he is at present professor emeritus, department of geology, University of Wisconsin at Madison, where he has been teaching since 1902.

George Huber of Montana School of Mines, Butte, has received the Viola Vestal Coulter Award of $200.00 for 1955-56.

F. H. Stewart, vice president and general manager of the Southwest Potash Corp. (a subsidiary of the American Metal Co.) Carlsbad, N. M., has been appointed manager, Mining and Exploration Dept., The American Metal Co., with headquarters in New York. **W. Aubrey Smith,** assistant general manager, Southwest Potash Corp., succeeds Mr. Stewart as general manager.

LLOYD R. BARNWELL

Lloyd R. Barnwell is now associated with Allen Engineering Assoc. Inc., San Francisco, as mining and metallurgical consultant. Prior to this he was operations supervisor, Permanente Metals Corp., and assistant plant engineer for International Minerals & Chemical Corp. in San Jose, Calif.

Theodore D. Tiemann is now director, Alumina Research Laboratory, Reynolds Metal Co., Hurricane Creek Plant, Bauxite, Ark.

DONALD A. RAMSAY

Donald A. Ramsay, formerly engineer with McNeil Robinson, County Surveyor, Birmingham, is now associated with Sverdrup & Parcel Engineering Co., St. Louis. Mr. Ramsay is in charge of field engineering parties on design surveys for roads in Cassville area, Rock Dam and Roaring River in Missouri.

Torgus Oaas, assistant general superintendent of mines, The Anaconda Co., Butte, Mont., has retired. Mr. Oaas had been connected with the company in various capacities for more than 43 years. He went to Bergen, Norway in 1907 and attended the Bergen Trade School. He attended Notre Dame University from 1910 to 1912 and then worked for six months at the U. S. Steel mill in Gary, Ind. Later he returned to Butte and worked as a miner at the Tramway mine. During the next few years he worked as shift boss, safety engineer and foreman, and in 1928 he was made assistant superintendent of mines.

William Embry Wrather, director of the U. S. Geological Survey since 1943, has retired. Mr. Wrather is well known for his pioneering applications of geology to the underground search for oil and for his war time service with the Metals and Minerals Div. of the Board of Economic Warfare. In 1950 he was awarded the AIME Anthony F. Lucas Petroleum Medal, and in 1945 the 50th John Fritz Medal, the highest engineering award in this country.

Robert N. Breckenridge is mine superintendent, Eureka Corp. Ltd., Eureka, Nev.

Donald J. Drinkwater has been appointed assistant manager, Marcy Mill Div., The Mine & Smelter Supply Co., Denver, Colo. Mr. Drinkwater was associated with U. S. Steel Corp. and the Ozark-Mohoning Mining Co. prior to joining Mine & Smelter as a metallurgical engineer in 1946. He graduated from the Colorado School of Mines with a degree of engineer of mining.

Howard W. Miller has been appointed superintendent, zinc mining operations, The New Jersey Zinc Co., Jefferson City, Tenn. Mr. Miller joined the company in 1946 as a geologist, exploration unit, Austinville, Virginia mine. He was transferred to Jefferson City in 1949.

Joseph P. McDaniel is superintendent of exploration, Loma Uranium Corp., Denver. Mr. McDaniel was with the Resurrection Mining Co., Leadville, Colo.

Maurice H. Brady, Sabre Uranium Corp., Grand Junction, Colo., has been transferred from the Utah-Colorado Div. Mr. Brady is now engineer in charge of the exploration work in the Grants, N. M. area.

B. M. Andreas has been appointed manager of mines of the Minnesota Div., M. A. Hanna Co. **Robert C. Wallace** is assistant manager of mines. **Robert B. Crist** is general superintendent, Minnesota Div., and will be located in the Hibbing, Minn. office. **Fred E. Dyson** succeeds Mr. Crist as superintendent of the Butler properties at Cooley, Minn.

David J. W. Frazer is process metallurgist, Automatic Auto Transmission, Marvel-Schebler Div., Borg-Warner Corp., Decatur, Ill. He was with International Rolling Mill Production Corp., Dolton, Ill.

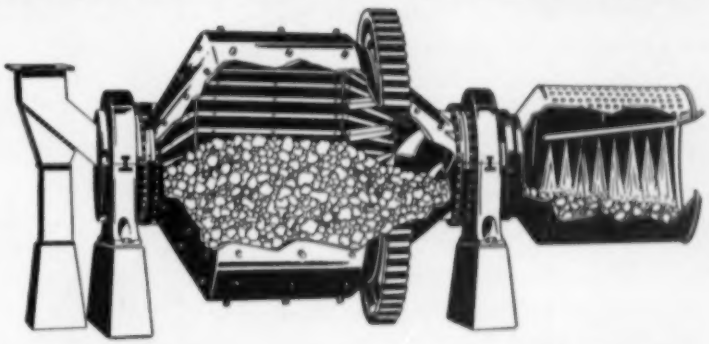
Harmon C. Ray is now special field engineer, Island Creek Coal Sales Co., Cleveland. Mr. Ray was with Cleveland-Cliffs Iron Co., Cleveland.

R. W. Whitney, M. A. Hanna Co., Hibbing, Minn., is moving to the company's office in Cleveland, where he will be general manager of mines. Mr. Whitney joined the company in 1946 as manager of the Minnesota Div. Prior to that he was with Butler Bros. on the Mesabi Iron Range.

Matthew P. Nackowski has resigned as associate professor of geology, Geology Dept., Missouri School of Mines, Rolla. Mr. Nackowski has accepted an appointment as associate professor of mining geology, Dept. of Mining and Geological Engineering, Univ. of Utah, Salt Lake City.

Philip R. Hammond, acting manager, Hercules Powder Co., Pittsburgh office, has been appointed assistant sales manager and transferred to the Wilmington office. **Thomas E. Brown,** manager of the Hazelton, Pa., sales office, has been appointed manager of the Pittsburgh office.

HARDINGE SCRUBBERS FOR PIT-RUN MATERIALS



Large diameter, short length trunnions permit chute feeding of unsized ore and rock at rates up to 600 tons per hour.
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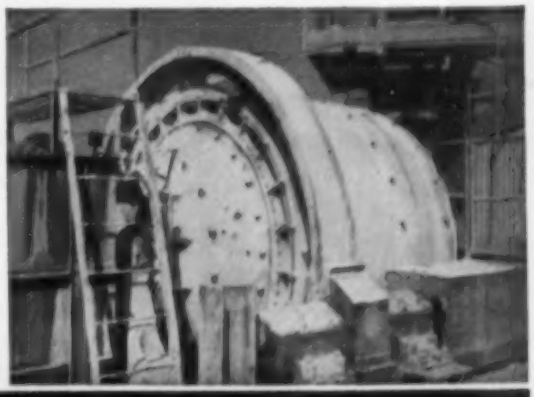

View of a 10′ x 66″ Hardinge Scrubber cleaning crushed dolomite in a California plant.

Bulletin 37-A-2

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W. J. CHERONES

W. J. Cherones was appointed to the newly created position of manager of electrical engineering at Harnischfeger Corp., Milwaukee. A member of the firm since 1945, Mr. Cherones has been chief engineer, electrical applications division, since 1950. He is a graduate of the University of Illinois and a member of AIEE and the Wisconsin Society of Professional Engineers.

R. B. Fuller has been appointed special consultant to the Phosphate Minerals Div., International Minerals & Chemical Corp. Mr. Fuller has been associated with International for nearly 30 years. Since 1953 he has been assistant to the vice president in charge of the Phosphate Minerals Div. Prior to that he was manager of the Florida Phosphate Dept. In 1951 Mr. Fuller received an honorary degree of doctor of laws from Florida Southern College. He is a director of Florida Power Corp., the First State Bank of Lakeland, Fla., and the Bank of Mulberry, Fla.

Allen D. Look, chief, Arizona Section Accident Prevention Branch, U. S. Bureau of Mines, Phoenix, Ariz., is now district supervisor, Accident Prevention and Health, USBM, Duluth.

Sherman A. White and **Floyd E. Albertson** have been appointed assistant general managers, Nickel Processing Corp. (a subsidiary of National Lead Co.) in Nicaro, Cuba. Mr. White joined National Lead in 1948 and went to Cuba as general superintendent of mines for Nickel Processing Corp. in 1953. Mr. Albertson came to National Lead in 1953 as assistant mill superintendent at the company's MacIntyre Development, Tahawus, N. Y. Prior to that he was mill superintendent for Cia. Minera de Oruro in Bolivia.

D. J. Ottley, general mill foreman, Cerro de Pasco Corp., Morococha, Peru, is now general foreman in charge of the company's new pilot plant in Cerro de Pasco.

W. B. STEPHENSON

Wm. B. Stephenson has been elected president of The Allen-Sherman-Hoff Pump Co., succeeding **Homer E. Allen,** who will serve as chairman of the board of directors. Well known in the mining field, in the U. S. and abroad, as an authority on the pumping of abrasives and corrosives. Mr. Stephenson has been identified with the company since his graduation from Pennsylvania State University in 1933. It was while he was vice president that the series of interchangeable Hydroseal and Centriseal rubber-lined slurry pumps were introduced. An active member of AIME, Mr. Stephenson is 1956 Program Chairman for MBD.

Mr. Stephenson is succeeded as vice president by **Fred S. Stow,** who will continue as chief engineer. **Donald G. Ashe** has been named assistant sales manager of the home office in Wynnewood, Pa. **Wm. M. Walters** is central district sales manager with headquarters at the company's Chicago office, and **Robert M. Jensen** has been placed in charge of the company's New York office.

EDWIN SWEETMAN

Edwin Sweetman, mining engineer of Green Bay, Wisc., is a member of the partnership which recently sold the Hotel Nicollet in Minneapolis for $4 million.

HAROLD B. EWOLDT

Harold B. Ewoldt has been appointed assistant to the president for mining matters at Cerro de Pasco Corp., N. Y. Prior to joining the company, Mr. Ewoldt served as vice president of Copper Range Co. and as manager of the zinc division of Calumet & Hecla Inc. After graduation from South Dakota School of Mines in 1930, he joined the U. S. Dept. of Agriculture as a mineral examiner for the Forestry Service, and later worked at USBM.

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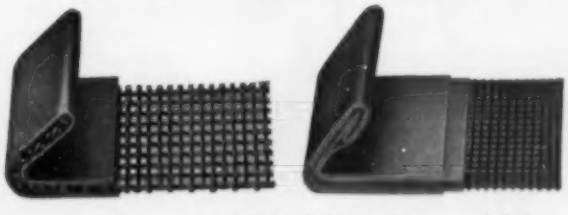
Tyler Screen Sections
for All Makes
of Screening Machines!



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Tyler rugged, accurately-applied hook-strips make possible stretching and maintaining the screens at drum-head tension, which is essential for successful screening and long screen life.

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Canadian Plant—St. Catharines, Ontario, Canada

OBITUARIES

Rev. James B. Macelwane, S.J.

An Appreciation by
Fred J. Meek

Rev. James B. Macelwane (Member 1937), *Father Macelwane* to his many friends in the Institute, was born on a farm near Port Clinton, Ohio, Sept. 28, 1883. His early years were those of a midwestern farm boy, followed by attendance at St. John's College High School in Toledo; St. Stamislaws branch of John Carroll University, Cleveland; St Louis University; University of California; and study at the University of Giessen in Germany. His degrees include: A.M., M.S., Ph.D., Sc.D., (HON.), LL.D. (HON.).

He started his university teaching career at St. Louis University in 1912, where he spent the greatest part of his life, with interludes of three years as assistant professor of physics at the University of California, foreign study and technical missions. At the University of California he organized graduate study in geophysics and had charge of the university seismograph, as well as that of Lick Observatory on Mt. Hamilton near San Jose, Calif.

Returning to St. Louis University, he became professor of geophysics and director of the new dept. of geophysics, the first such department in the western hemisphere. He was further honored by the office of dean of the Graduate School, followed by appointment as dean of the Institute of Geophysical Technology, now broadened into the Institute of Technology, St. Louis University. His reports to the press on earthquake recordings over many years, made his name familiar to all in St. Louis.

Father Macelwane was a man willing to take responsibilities and to serve every worthwhile cause. Though known to most of us by his scientific endeavors, he was a fine example to mankind of the Jesuit Order which he entered in 1903, becoming ordained as a priest in 1918. He had a quality of friendliness and good humor that made even those of us of the Protestant faith feel equally of a spirit with him and proud to have his companionship. The St. Louis Section knew it could depend on his faithful attendance, cooperation on any occasion of need and his maintenance of a strong student chapter year in and year out.

That he was likewise appreciated by his fellow townsmen was illustrated when more than a thousand people called to congratulate him recently, on the occasion of his completion of 50 years in the Society of Jesus. He was a prolific writer and had taken a leading part in the conduct of affairs in scientific societies too numerous to mention, accepting on occasion commissions of contact and cooperation with many foreign scientific conclaves and events.

Father Macelwane's final illness and death on Feb. 15, 1956, occurred as he was about to be honored for his achievements, with the privilege of presenting the Jackling Lecture for the AIME Mining, Geology, and Geophysics Div. at the Annual Meeting, an honor he appreciated greatly. His absence from AIME councils, both local and national, will be a great loss and his friendship sorely missed.

Necrology

Date Elected	Name	Date of Death
1933	Walter D. Abel	Aug. 17, 1955
1906	Arthur Howe Carpenter	Mar 20, 1956
1899	T. R. Drummond	Feb. 28, 1956
1900	Daniel C. Jackling	Mar. 13, 1956
1950	Harry E. Kamprath	Mar. 15, 1956
1920	H. S. Lyne	Feb. 24, 1956
1912	Arthur G. McKee	Feb. 19, 1956
1931	H. C. Miller	Dec. 18, 1955
1956	William F. Mosier	Dec. 26, 1955
1913	Russell B. Paul	Mar. 5, 1956
1954	Hollis P. Sawyer	Unknown
1953	Frank W. Stedman	Oct. 6, 1955
1941	Watson Storey	October 1955
1952	William R. Stout	Mar. 10, 1956
1955	Ross M. Stuntz, Jr.	Jan. 18, 1956
1952	Allan Richard Volmar	July 16, 1955
1910	Thor Warner	Jan. 20, 1956
1948	David M. Wright	Mar. 11, 1956

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Charles W. Merrill

An Appreciation by
Herbert S. Shuey

Charles W. Merrill (Legion of Honor Member 1892), internationally known San Francisco metallurgical engineer, died Feb. 6, in Honolulu.

Mr. Merrill, holder of the James Douglas Medal of the AIME and a former regent of the University of California, his alma mater, was in the midst of one of his frequent visits to Hawaii, and had been ill only three days. He was 86.

Born in Concord, New Hampshire, Dec. 21, 1869, Mr. Merrill was brought to San Francisco as an infant by his parents; and, although his work in the metallurgical field took him over the globe, he had maintained residence in the San Francisco Bay Area all his life.

A graduate of the University of California with the class of 1891, he immediately entered upon the career which was to bring him international fame for his development, design, installation, and operation of numerous gold extraction plants in the U. S., Canada, and Mexico. These plants were based on his own patents involving various processes for the recovery of gold from complex ores. At various times Mr. Merrill held more than 25 patents on metallurgical processes and their application, many of which proved so successful as to be ultimately adopted by most of the large gold and silver mines throughout the world.

Upon graduation from college, Mr. Merrill spent a short time as assistant to Dr. George B. Becker, Chief of the U. S. Geological Survey. Later that year (1891) he met and went to work for the eminent metallurgist, Alexis Janin, whose brother—the mining engineer Louis Janin—introduced him to another mining engineer—Herbert Hoover. Mr. Merrill spent three years with Alexis Janin in San Francisco, working in the laboratory on metallurgical problems.

Having prepared his graduation thesis on the cyanide process for recovering gold from its ores, Mr. Merrill was anxious to install a plant for commercial application; and, after leaving the employ of Alexis Janin, struck out for himself. He went to Bodie, Calif., and, after studying the tailings of the Standard Consolidated Mine there, proposed to its superintendent, Mr. Thomas H. Leggett, the erection of an experimental plant. Mr. Leggett agreed, and the plant proved successful.

Mr. Merrill then spent about two years at the rich Harqua Hala Mine in Arizona. There he persuaded its superintendent, Robert M. Raymond, to let him install a cyanide plant, which likewise proved successful.

Shortly after completing this project, Mr. Merrill met two mining engineer friends—Messrs. Wartenweiler and Bratnober—who told him of the operations of the Montana Mining Co. at Marysville, Mont. Mr. Merrill went there and induced its manager, A. T. Bayliss, to install an experimental plant. When this plant had demonstrated that the mine's tailings could be profitably reworked, a large operating plant was installed, of which Mr. Merrill had charge as resident engineer, continuing as consultant for several years.

In 1898 Mr. Merrill, after extended study of the operations of the famed Homestake Mine at Lead, S. D., entered into a contract with Homestake Mining Co. to install a cyanide plant for recovery of gold from tailings, and spent ten years in directing this highly successful operation—an operation which resulted in recovering $2000 worth of gold per day which previously had been going to waste in the slimes. It was the success of this large operation which first brought Mr. Merrill worldwide recognition and acclaim.

After his ten years at Homestake, Mr. Merrill returned to San Francisco and established his office there. This was in 1908. Two years later, he organized The Merrill Co., to make his processes, methods, and equipment available on a world basis. Later, he organized a number of subsidiary companies to handle the manufacture and distribution of various technical equipment and products developed by himself or his associates. He was active in these companies for many years, and was board chairman of The Merrill Co. at the time of his death.

In the field of public service Mr. Merrill was active nationally and locally. During World War I, at the request of his lifelong friend Herbert Hoover, Sr., he accepted assignment in Washington as Chief of the Div. of Collateral Commodities in the Food Administration, serving in that capacity until the end of the war. During World War II his various enterprises were called upon by the Armed Services to extend and vary their endeavors and processes as their part in aiding the war effort.

Mr. Merrill was a vice president of the AIME in 1924, and in that year was awarded the Institute's James Douglas Medal for "distinguished international metallurgical achievement." In 1924 and 1925 he was a member of the Board of Regents of the University of California, serving also as president of the Alumni Assn. In the city of San Francisco he served as chairman of the Community Chest from 1925 to 1927, and later was chairman of the California State Mining Board and the Minerals Committee of the State Chamber of Commerce.

In 1940 Mr. Merrill won an award from the NAM for "distinguished achievement in the field of science and invention, which has advanced the American standard of living." In recent years he held various posts with the U. S. Bureau of Mines.

Mr. Merrill was a member of the University, Engineers,' Pacific Union, and Commonwealth Clubs of San Francisco; the Mining and Metallurgical Society of London; the Australasian Institute of Mining Engineers; and the Chemical Metallurgical, and Mining Soc. of S. Africa.

Joseph H. Hedges (Member 1907) died Dec. 1, 1955 in Washington, D. C., after a short illness. Before he retired in 1955, Mr. Hedges was staff adviser to the Division of Minerals, USBM, Washington, D. C. He was born in Lansing, Mich., in 1882, and received a degree in mechanical engineering from Michigan State College, and in mining engineering from the Michigan College of Mining and Technology. Before joining the USBM in 1926 Mr. Hedges worked for several mining companies in the U. S., Canada, and Mexico. From 1926 to 1940 he was special assistant to the director USBM in Washington, D. C. He then transferred to College Park, Md., as chief of the Eastern Strategic Minerals Section. In 1952 he became chief of the Tucson, Ariz., branch of the Mining Division, and took charge of all USBM mineral investigations in Arizona, New Mexico, and Texas. Later he was superintendent, Southeast Experiment Station, Tucson. From 1943 to 1945 Mr. Hedges supervised work on the San Manuel copper project, Pinal County, Ariz. He was recalled to Washington in 1949 as chief of the USBM Minerals Division, and in 1952 became a special assistant to the director. In 1953 he received the Distinguished Service Award, the Interior Dept.'s highest honor. Mr. Hedges was the author of many USBM publications on mining and minerals technology.

Clare Edwin Nighman (Member 1911) died Jan. 11, 1956, after a long illness. Mr. Nighman was chief, European Div., Foreign Minerals Region, USBM, before retiring in 1953. At that time he was awarded the Dept. of Interior Silver Medal for meritorious service. He was born in Canton, Ohio, in 1887. Mr. Nighman graduated from Columbia University in 1909 with a B.S. in geology and mining. Later he returned to Columbia University and received a degree in mining engineering. From 1909 to 1924 Mr. Nighman worked for several mining companies. He then went to The Anaconda Co. at Butte, Mont., and later was transferred to Idaho in charge of the development of phosphate rock production. Mr. Nighman also worked for Anaconda in Poland as manager of coal mining operations. He returned to the U. S. in 1934 and became a consulting engineer. Mr. Nighman entered Government service in 1938 as a consultant to the Work Progress Administration, and transferred to the USBM in 1940.

George W. Bryant

An Appreciation by
Alvin Carpenter

George W. Bryant (Legion of Honor Member 1904) died in Los Angeles Dec. 5, 1955, after a gradual loss of physical strength during a period of several months. He was 92 years of age. Mr. Bryant became a Life Member of AIME in 1905.

Despite the fact that he had been totally blind during the last eight years of his life, he never appeared depressed, was always patient and cheerful, with an alert mind and a keen interest in the world of mining and metallurgy. Through radio and recordings, he kept abreast of the times and took part in discussions of developments in the mining field.

Born in Boston, Mr. Bryant went to Mexico in his early twenties as an accountant with one of the early English companies that had attempted a revival of the old Spanish mines of that country and had properties in the state of Guanajuato. At that time the company had weathered a long period of revolutions and was badly outdated in its mining and milling methods. It did not long survive the turn of the century, but in the meantime Mr. Bryant had, in his 12 years of experience with the company, arrived at the position of manager of several of the company's mines. He had acquired a wide experience and insight into mining and the mining possibilities of the area, the extent and character of the ores, as well as the ownership and titles of the mining properties. All of this was to prove of value in the years immediately following.

It was at this time that a new era dawned on Mexico for its languishing mining industry. The grinding of ore in Chile mills and with arrastres was being replaced with stamp mills and ball mills; the old patio process and panamalgamation was being succeeded by the cyanide process.

Mr. Bryant's systematic and careful correlation of engineering data was of the highest order. There was no one more liked and respected by financiers, investors, and engineers.

George Arnold Parker (Member 1946) died Oct. 26, 1955. Mr. Parker was manager, Hercules Powder Co., Pittsburgh. He was born in New Castle, Ala., in 1904, and graduated from Alabama Polytechnic Institute with a B.S. degree in 1927. Mr. Parker then joined Birmingham Electric Co. as a junior engineer, and later became assistant station master. From 1929 to 1935 he worked for New Castle Co., and was with the TVA Safety Dept. for a year. He joined the Engineering Dept., Hercules Powder Co. in 1936.

Harry Erhard Kamprath (Member 1934) died Mar. 15, 1956. He was assistant milling engineer, American Smelting & Refining Co., Salt Lake City. Born in Freiberg, Saxony, Germany, in 1902, Mr. Kamprath received his degree of Mining Engineer at the Freiberg School of Mines in 1927. Early in 1930, as winner of an all-European scholarship he was granted a fellowship at Montana School of Mines where he received his M.S. in metallurgical engineering in 1931. From 1931 to 1936 he worked as mine and mill superintendent for Glengarry Mining Co. in Cooke, Mont., and as metallurgical engineer with Montana Consolidated Mines Corp. In 1937 he returned to Germany for 12 years, working as mining and metallurgical engineer with various companies, and finally with the U. S. Army Corps in Frankfurt. From December 1949 to December 1952 Mr. Kamprath was mill superintendent of the Paragsha concentrator for Cerro de Pasco Corp. at Cerro de Pasco, Peru. He then returned to the U. S. where he became associated with American Smelting & Refining Co.

Harold V. Trask (Member 1951) died at Lakeside Hospital, Cleveland, on Dec. 23, 1955. Mr. Trask was chief metallurgist, Cleveland-Cliffs Iron Co., Cleveland office. Born in Johnston, N. D., in 1915, he graduated from the University of North Dakota in 1937 with a B.S. Mr. Trask held a research fellowship at the University of Utah and received an M.S. in 1938. He was employed as a mining engineer at Mountain City Copper Co. of Utah, and in 1940 became a metallurgical engineer at Talache Mines Inc. of Utah. Among the companies in the iron and steel and allied industries with which Mr. Trask was associated were Butler Brothers, Minn., and Buel Metals Co., where he was manager. From 1950 to 1952, Mr. Trask was affiliated with Oglebay, Norton Co., assigned as a technical advisor to Armco Steel Corp., Ashland, Ky. He joined Cleveland-Cliffs as a metallurgist in 1952. Mr. Trask was a member of the program committee, AIME Committee on Blast Furnace, Coke Oven, and Raw Materials.

Archimedes B. Woodward, Jr. (Member 1944) died Nov. 13, 1955. Mr. Woodward was a consulting engineer and partner in the New Mexico Engineering Co., Albuquerque, N. M. He was born in San Francisco in 1904. After attending the University of California, he worked for United Verde Copper Co. In 1929 he returned to the university as an engineer, surveyor, and draftsman in the Engineers office. Mr. Woodward then spent several years working in various mines in the U. S. He joined New Mexico Engineering Co. in 1952.

Robert Emmett Dye (Member 1915) died Nov. 15, 1955 at his home in San Antonio, Texas. Mr. Dye was born in South Dakota in 1888. He graduated from the Missouri School of Mines, Rolla, with a B.S., and received an E.M. from the university some years later. In 1912 Mr. Dye went to Canada and worked for Buffalo Mines Ltd., Cobalt. Later he was transferred to the company's Teck-Hughes mine, Kirkland Lake, as mill superintendent. In 1919 Mr. Dye returned to Cobalt to take charge of the Northern Customs Concentrator and other operations. From 1923 to 1934 he was with Vipond Consolidated Mines Ltd., Timmins. He then went to Dome Mines Ltd., South Porcupine, as mine superintendent, becoming vice president and general manager in 1945. Mr. Dye retired in 1953, but was retained as vice president and consultant. He had been a member of the Canadian Institute of Mining and Metallurgy since 1916 and served as a vice president of the CIM in 1941-1942. Mr. Dye was also a past-president of the Ontario Mining Assn.

Ernest Russell Dickie (Member 1945) died Dec. 14, 1955. Mr. Dickie was treasurer and general manager, Bagdad Copper Corp., Bagdad, Ariz. He was born in Independence, Colo., in 1902. Mr. Dickie began work as a truck driver for United Eastern Mining Co., Oatman, Ariz., and later for Topock Oil Co. From 1924 to 1931 he was with United Verde Copper Co., Jerome, Ariz., as miner and mechanic. Mr. Dickie then joined East Vulture Mining Co., Wickenburg, Ariz., as general superintendent and master mechanic, and became president and general manager of the company in 1936. He had been general manager of the Bagdad Copper Corp. for many years.

Cecil H. Brehaut (Member 1955) died Jan. 14, 1956 in Montreal, after being struck by a car while crossing a street. Mr. Brehaut was operating engineer, Canadian Johns-Manville Co., Asbestos, Que. He was born at Guernsey Cove, Prince Edward Island, Nova Scotia, in 1901. After graduating from Nova Scotia Technical College with a B.S. in mining engineering, Mr. Brehaut joined Granby Consolidated Mining & Smelting Co., Anyox, B.C., in 1930. From 1935 to 1937 he was with Seal Harbour Gold Mines in Nova Scotia. Mr. Brehaut then returned to the Granby Co. as chief engineer at Copper Mountain, B.C. In 1944 he went to Beattie Gold Mines at Duparquet, Que., and later worked for Lamaque Gold Mines Ltd. at Bourlamaque. He joined the Canadian Johns-Manville Co. in 1947. Mr. Brehaut was a member of the CIM and served as secretary of Sherbrooke Branch in 1953.

Arthur Howe Carpenter

An Appreciation by
Malcolm H. Carpenter

Arthur Howe Carpenter (Legion of Honor Member 1905) died Mar. 20, 1956 in Middleport, Ohio, after an extended illness.

Born in Georgetown, Colo., Oct. 19, 1877, Mr. Carpenter received his B.A. and M.A. (1914) from Ohio University. He also studied at Northwestern University. Among the various positions he held in the field of metallurgy were: assayer, chemist, prospector, executive, and partner, during the early 1900's in the West. From 1920-28 he was assistant professor of metallurgy at the Illinois Institute of Technology, Chicago, where he later became associate professor and head of the metallurgy div., and in 1944, professor emeritus. He also lectured at the Institute on geology, astronomy, and meteorology.

To Mr. Carpenter's credit is the invention of a process for covering pipe with lead. An astronomer, he manufactured the telescope used by Elgin Observatory for the Areturus Ceremony at the Century of Progress Exposition in Chicago, 1933-1934. The telescope, made as a memorial to his father, Dr. Franklin Carpenter and also dedicated to Mrs. Carpenter's father, D. J. Evans, was later presented to Ohio University.

Listed in *Who's Who in America*, 1954-1955, Mr. Carpenter was associated with many companies including: Clear Creek Mining & Reduction Co., Golden, Colo., where he was superintendent from 1901 to 1903; Takilma Mining & Smelting Co., Takilma, Ore., as general manager; and American Smelting & Refining Co., Denver, as chief chemist.

He was a member of the Illinois Academy of Science, Chicago Astronomy Society, ASTM, AAAS, Phi Lambda Upsilon, Alpha Chi Sigma, Pi Gamma Mu, Sons of the American Revolution, and the Masons. Surviving are his wife; two daughters, Mrs. Dutton, and Mrs. S. L. McCarthy; one son, Franklin; three grandchildren; three brothers; and a sister.

George Harold Anderson

An Appreciation by
J. L. Gillson

George Harold Anderson (Member 1939), who was born in Elko, Nev., on Nov. 21, 1893, passed away in New York Feb. 16, 1956. He took his A.B. and A.M. degrees at Stanford University in 1917 and 1920, respectively, later studied at Harvard and at the University of Colorado, and received his Doctor's degree in geology from the California Institute of Technology in 1933. Upon graduation from Stanford he was elected to Sigma Xi and Phi Beta Kappa.

Dr. Anderson, who was intensely interested in the problems of granitization, wrote his doctor's thesis on such phenomena in the northern Inyo range of California. This thesis was published in the *Bulletin of the Geological Society of America*, vol. 48, pp. 1-74, 1937.

However, Dr. Anderson's active mind later turned to more commercial matters. For three years, he was in charge of exploration for titanium ores in the San Gabriel mountains, north of Los Angeles, for the du Pont de Nemours Co., and then he joined the staff of the Texas Power & Light Co. at Dallas. That company had devoted their energies to developing the natural resources of Texas and especially to starting a blast furnace and steel plant to supply steel for the growing oil industry of the state.

Dr. Anderson was in charge of the development of the east Texas iron ore fields which had been examined and condemned by a generation of earlier geologists. He lived to see the establishment of a great steel industry based on those ore deposits.

During the Second World War, he spent a great deal of time and effort in attempting to get the War Production Board to permit this steel development. After the war, the management of the Texas Power & Light Co. saw that the great supplies of natural gas in the state were not sufficient for the long range future of Texas expansion. Dr. Anderson initiated an investigation that led the Aluminum Co. of America to build their smelter at Rockdale, Texas, using lignite as fuel.

His achievements for the Texas Power & Light Co. resulted in his appointment as assistant to the president, in which capacity he commanded the respect and love of his associates. During his later years, Dr. Anderson devoted much of his time to bringing other major industries to Texas. The native of Nevada became an enthusiastic Texan.

He was a Fellow of the Geological Society of America, the Mineralogical Society of America, and a member of the Society of Economic Geologists, American Assn. of Petroleum Geologists, and AIME.

Dr. Anderson is survived by his wife and three children, two of whom are doctors of medicine.

Royal Sheppard Handy (Member 1916) died Dec. 24, 1955. Mr. Handy was a consulting metallurgist at Santa Rosa, Calif. He was born in Loveland, Colo., in 1880. After attending the University of California, he joined Red Boy Consolidated Mining Co., Granite, Ore., as assayer, bookkeeper, and surveyor. Later he was smelter representative and yard foreman at The Bunker Hill Co., becoming mill superintendent in 1909.

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The Institute desires to extend its privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

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Coming Events

May 17-18, Rocky Mountain Petroleum Sections, joint meeting, Gladstone Hotel, Casper, Wyo.

May 18-19, AIME, Central Appalachian Section, spring meeting, Martha Washington Inn, Abington, Va.

May 19, AIME, Colorado MBD Subsection, annual meeting with **Pikes Peak Section**, Broadmoor Hotel, Colorado Springs, Colo.

May 26, AIME, Adirondack Section, tour of Power and Seaway Projects, Massena, N. Y.

June 7, AIME, Utah Uranium Subsection, 7:30 p.m., Arches Cafe, Moab, Utah.

June 16-25, Fifth World Power Conference, Vienna.

June 17-22, American Society for Testing Materials, annual meeting, Hotel Chalfonte-Haddon Hall, Atlantic City, N. J.

June 23, AIME, Adirondack Section, Mineville, N. Y.

June 28, AIME, Adirondack Section, Gouverneur, N. Y.

Aug. 19-24, Sixth International Symposium on Combustion, Yale University, New Haven, Conn.

Aug. 23-25, National Council of State Boards of Engineering Examiners, Hotel Statler, Los Angeles.

Sept. 4-11, 20th International Geological Congress, Mexico City.

Sept. 26-28, AIME Rocky Mountain Minerals Conference, Salt Lake City. Technical papers will represent all branches, including the Petroleum Div.

Sept. 29-Oct. 1, Society of Exploration Geophysicists, 26th annual meeting, New Orleans.

Oct. 1-4, American Mining Congress, Mining Show, Shrine Auditorium, Los Angeles.

Oct. 8-10, AIME, Institute of Metals Div., Carter Hotel, Cleveland.

Oct. 8-12, Union of Pan American Assns. of Engineers, fourth convention, Mexico City, Mexico.

Oct. 14-17, AIME, Petroleum Branch, Biltmore Hotel, Los Angeles.

Oct. 22-24, ASA, 36th annual meeting, Hotel Roosevelt, New York.

Oct. 29-Nov. 1, Society of Exploration Geophysicists, annual meeting, New Orleans.

Nov. 1-3, Geological Society of America, Minneapolis.

Nov. 8-10, AIME, Northeastern Mining Branch Conference, Hershey, Pa. Lehigh Valley Section is host.

Dec. 5-7, AIME, Electric Furnace Steel Conference, Hotel Morrison, Chicago.

Feb. 24-28, 1957, AIME Annual Meeting, Roosevelt and Jung hotels, New Orleans.

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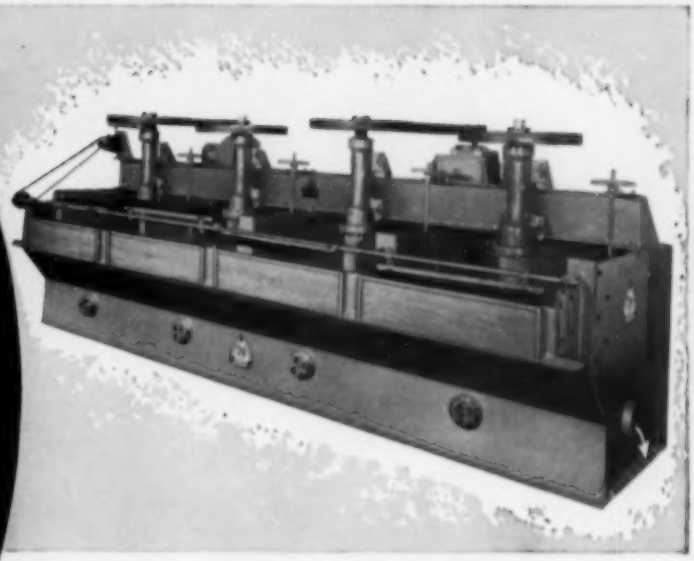

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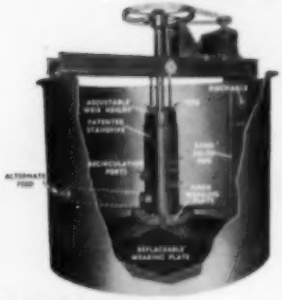

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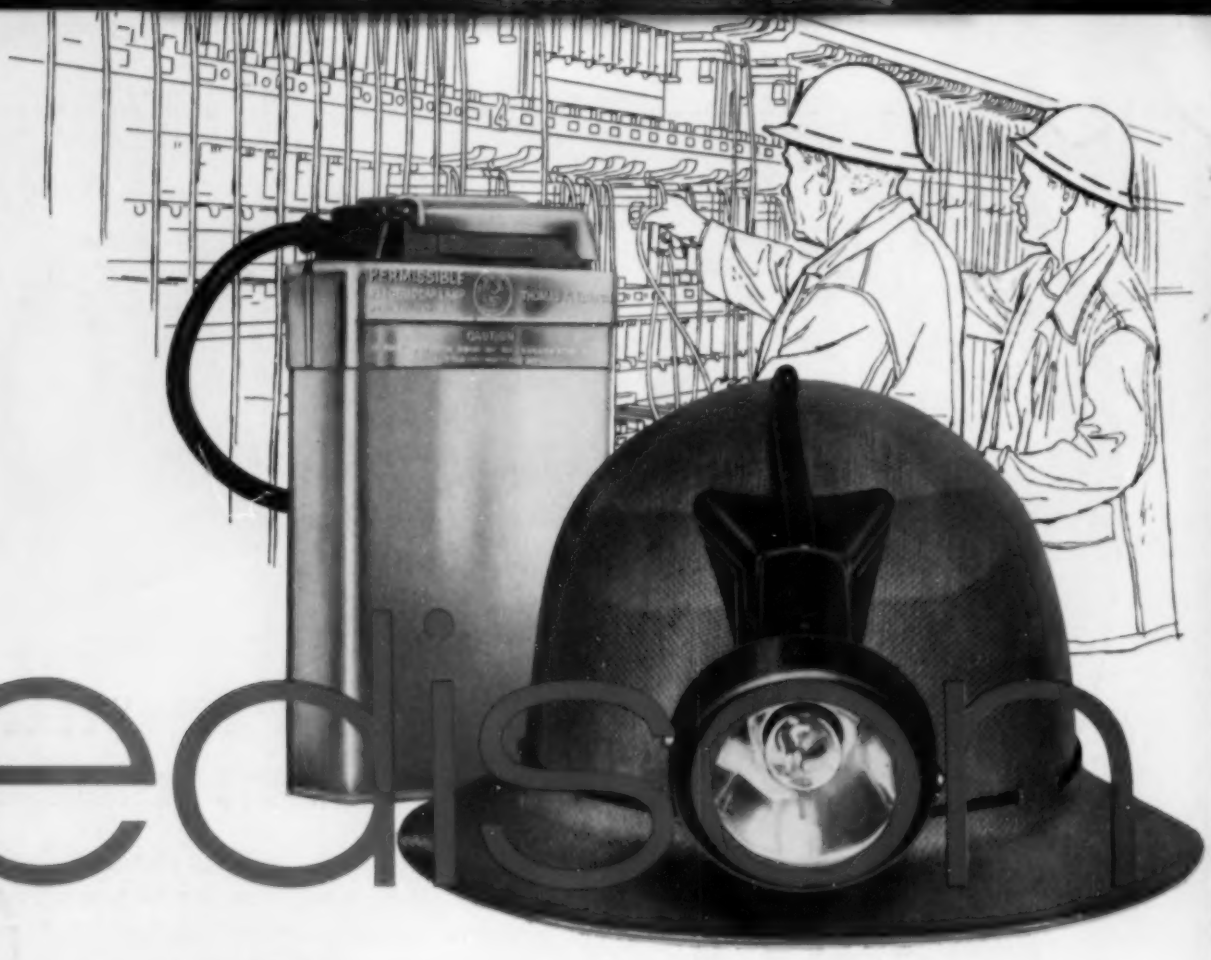

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